TN 295
.U4

THE LIBRARY OF CONGRESS

THE LIBRARY OF CONGRESS

IC 8933

Bureau of Mines Information Circular/1983

LIBRARY OF CONGRESS
NOV 4 1983
EXCHANGE COPY

Manganese Nodule Resources of Three Areas in the Northeast Pacific Ocean: With Proposed Mining-Beneficiation Systems and Costs

A Minerals Availability System Appraisal

By C. Thomas Hillman

UNITED STATES DEPARTMENT OF THE INTERIOR

Information Circular 8933

Manganese Nodule Resources of Three Areas in the Northeast Pacific Ocean: With Proposed Mining-Beneficiation Systems and Costs

A Minerals Availability System Appraisal

By C. Thomas Hillman

UNITED STATES DEPARTMENT OF THE INTERIOR
James G. Watt, Secretary

BUREAU OF MINES
Robert C. Horton, Director

As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally owned public lands and natural resources. This includes fostering the wisest use of our land and water resources, protecting our fish and wildlife, preserving the environmental and cultural values of our national parks and historical places, and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to assure that their development is in the best interests of all our people. The Department also has a major responsibility for American Indian reservation communities and for people who live in Island Territories under U.S. administration.

TN295
.U4
no.8933

This publication has been cataloged as follows:

Hillman, C. Thomas

Manganese nodule resources of three areas in the northeast Pacific Ocean.

(Information circular / United States Department of the Interior, Bureau of Mines ; 8933)

Bibliography: p. 38-40.

Supt. of Docs. no.: I 28.27:8933.

1. Manganese nodules—Pacific Ocean. 2. Ocean mining—Pacific Ocean. 3. Ore-dressing. I. Title. II. Series: Information circular (United States. Bureau of Mines) ; 8933.

TN295.U4 [TN490.M3] 622s [553.4'629] 83-600014

For sale by the Superintendent of Documents, U.S. Government Printing Office
Washington, D.C. 20402

CONTENTS

ILLUSTRATIONS

TABLES

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

bbl/d	barrel per day
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
cm/s	centimeter per second
d	day
° C	degree Celsius
dwt	deadweight ton
d/yr	day per year
g/cm^2	gram per square centimeter
g/cm^3	gram per cubic centimeter
g/L	gram per liter
h	hour
ha	hectare
h/d	hour per day
hp	horsepower
kg/cm^2	kilogram per square centimeter
kg/d	kilogram per day
kg/m^2	kilogram per square meter
km	kilometer
km^2	square kilometer
kn	knot
kW•h	kilowatt hour
m	meter
m^2	square meter
m^3	cubic meter
min	minute
mm	millimeter
m/s	meter per second
MW	megawatt
nmi	nautical mile
m^3/min	cubic meter per minute
pct	percent
t	metric ton
t/d	metric ton per day
t/h	metric ton per hour
t/km^2	metric ton per square kilometer
t/yr	metric ton per year
vol pct	volume percent
wt pct	weight percent
yr	year

MANGANESE NODULE RESOURCES OF THREE AREAS IN THE NORTHEAST PACIFIC OCEAN: WITH PROPOSED MINING-BENEFICIATION SYSTEMS AND COSTS

A Minerals Availability System Appraisal

By C. Thomas Hillman[1]

ABSTRACT

The practical concern of economic minability of large, high-grade manganese nodule deposits in the northeast Pacific Ocean is addressed in this Bureau of Mines report. Principal objectives are to (1) estimate tonnage and grade of deposits with significant potential and (2) describe and estimate profitability of operations designed to mine and process deposits with greatest apparent potential.

Analysis of data from over 800 ship stations identified three areas for detailed study. Average metal contents of these areas range from 1.30 to 1.45 wt pct nickel, 1.00 to 1.24 wt pct copper, 0.21 to 0.26 wt pct cobalt, and 26.8 to 27.8 wt pct manganese. Estimated recoverable nodule resources are 67.0, 66.9, and 148.8 million dry metric tons (t).

A system to mine, transport, and process nodules from the three sites is described and costed. Although hypothetical, the system utilizes hydraulic mining and Cuprion (Kennecott) processing, which have been successfully tested at pilot scale. Nickel, copper, and cobalt are the three primary products, but ferromanganese is a considered option.

Estimated capital requirements are approximately \$1.5 to \$1.7 billion for three-metal production. If ferromanganese were recovered, an additional investment of about \$130 million would be required. Operating costs range from \$71 to \$83 per dry metric ton of nodules without manganese, and from \$103 to \$123 per dry metric ton with ferromanganese. Discounted cash flow analyses predict low returns, ranging from 2.7 to 5.2 pct with ferromanganese and from 4.1 to 6.0 pct without.

---

[1]Physical scientist, Western Field Operations Center, Bureau of Mines, Spokane, WA.

INTRODUCTION

Deep ocean manganese nodule deposits, representing a very large source of the metals nickel, copper, cobalt, and manganese, are the subject of continuing international controversy. The central issue is ownership and control, because the largest and highest grade deposits generally are in the deep ocean beyond territorial limits. Despite controversy, a substantial amount of exploration has occurred during the last 10 to 15 yr.

Various mining consortia with U.S., Canadian, European, and Japanese partners have prospected large areas of the world's oceans, especially the region known as the northeast Pacific high-grade zone. This area extends from about 110° W to 160° W longitude and 5° N to 20° N latitude, and is presently the area of primary commercial interest. Indications are that many potential minesites have been discovered and explored to varying degrees (28).[2] Because of high exploration costs, uncertain political climate, and the competitive nature of the business, most information has been kept proprietary. This policy, although proper, fuels the ownership controversy, because there is a belief by many that manganese nodules represent a source of tremendous profit for those in a position to grasp it. The fact that large, high-grade deposits exist, however, does not guarantee profit can be attained from their exploitation.

While much has been written concerning nodule resources, conceptual mine-process systems, and economics, no published work is available which addresses the question of profitability of mining a specific nodule deposit. Therefore, the purposes of this report are to analyze available information on deposits in the northeast Pacific high-grade zone in an attempt to identify areas of high grade and abundance, and determine their minability and profitability based on existing technology and economics. A final purpose, irrespective of profitability, is to analyze the potential beneficial impacts nodule production could have on the U.S. supply of nickel, copper, cobalt, and manganese.

The study was performed as part of the Bureau of Mines minerals availability program to inventory and assess the availability of nonfuel minerals. The basis for this program consists of evaluations of individual deposits. Each deposit report normally includes geological and geographical descriptions, resource-reserve estimates, mining and beneficiation plans, and an economic analysis. These same elements are present in this report, which is in reality a specialized and expanded minerals availability deposit report.

Resource information was gathered during the past 5 yr, through Bureau of Mines grants to Scripps Institution of Oceanography and Washington State University, and by contacts with personnel of the U.S. Geological Survey (26-28). Much data were developed as a spinoff of project DOMES (Deep Ocean Mining Environmental Study) carried out by the National Oceanic and Atmospheric Administration (NOAA). DOMES was a detailed investigation of the deep ocean environment of three potential minesites (A, B, and C) and a determination of possible effects of nodule mining. Figure 1 shows the northeast Pacific high-grade zone and locations of the three DOMES sites.

---

[2]Underlined numbers in parentheses refer to items in the list of references preceding the appendixes.

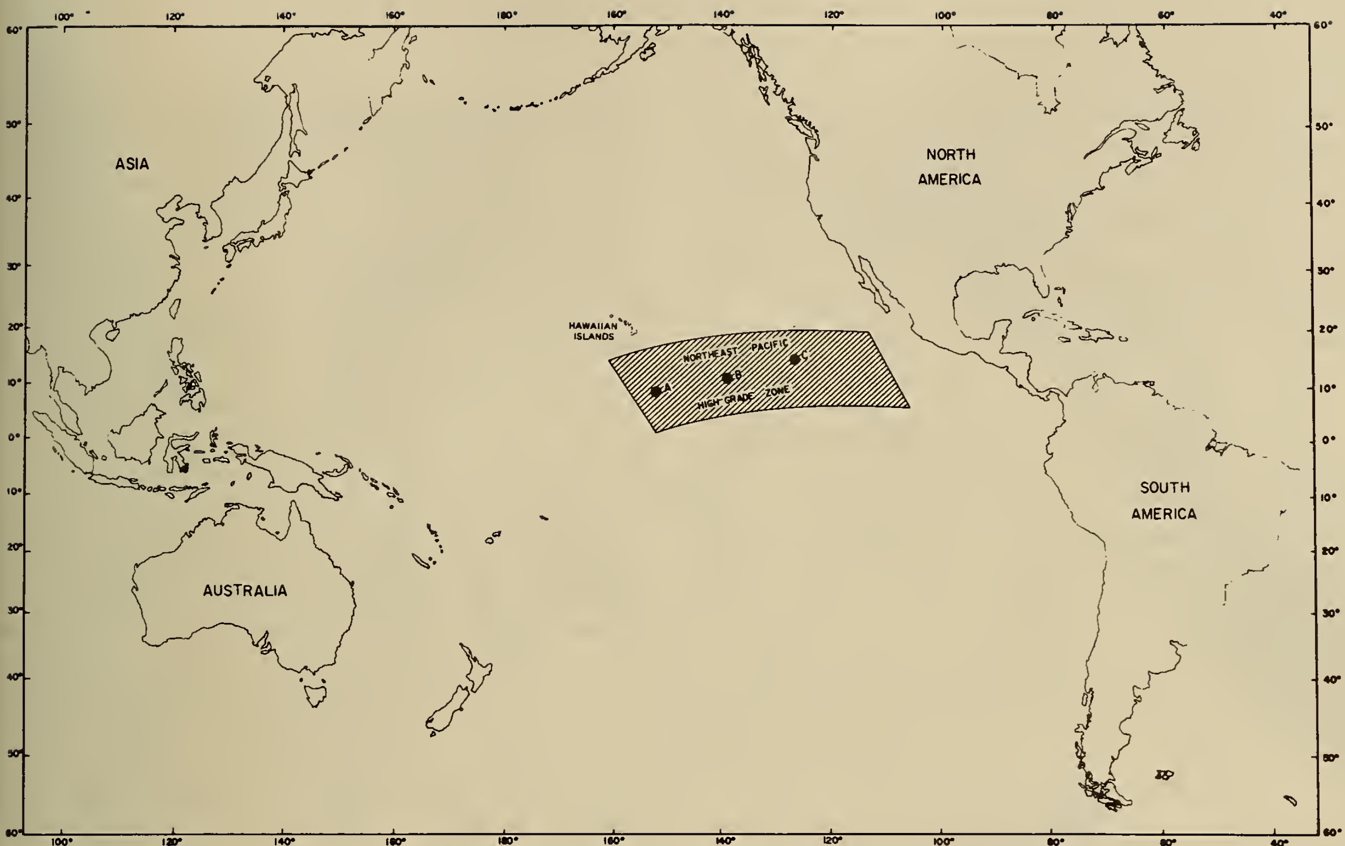

FIGURE 1. - Location of the northeast Pacific high-grade zone and DOMES Sites A, B, and C.

Resource data consist of nodule assays (dry weight percent), populations (percent of seafloor covered with nodules), and abundances (weight per unit area) from more than 800 ship stations situated in three broad areas encompassing DOMES Sites A, B, and C (Fig. 2). Because of apparent high grade and abundance, specific deposits within these areas are believed to have significant economic potential. A proposed system to mine, transport, and beneficiate nodules from these areas is described and costed.

Although no attempt was made to optimize either capacity or location of processing facility, the system represents a likely approach to recovery of these resources. A financial analysis, based on derived costs, was completed using a Bureau of Mines mine simulator computer program (MINSIM4). A discussion of results precedes a short analysis of the potential for reducing U.S. dependence on imports of nickel, cobalt, and manganese.

ACKNOWLEDGMENTS

The author expresses appreciation to Dr. John Flipse, ocean mining consultant, College Station, TX; Benjamin V. Andrews, ocean transportation consultant, Menlo Park, CA; and Dr. Francis Brown, process engineer, EIC Corp., Newton, MA; who provided technical and cost information.

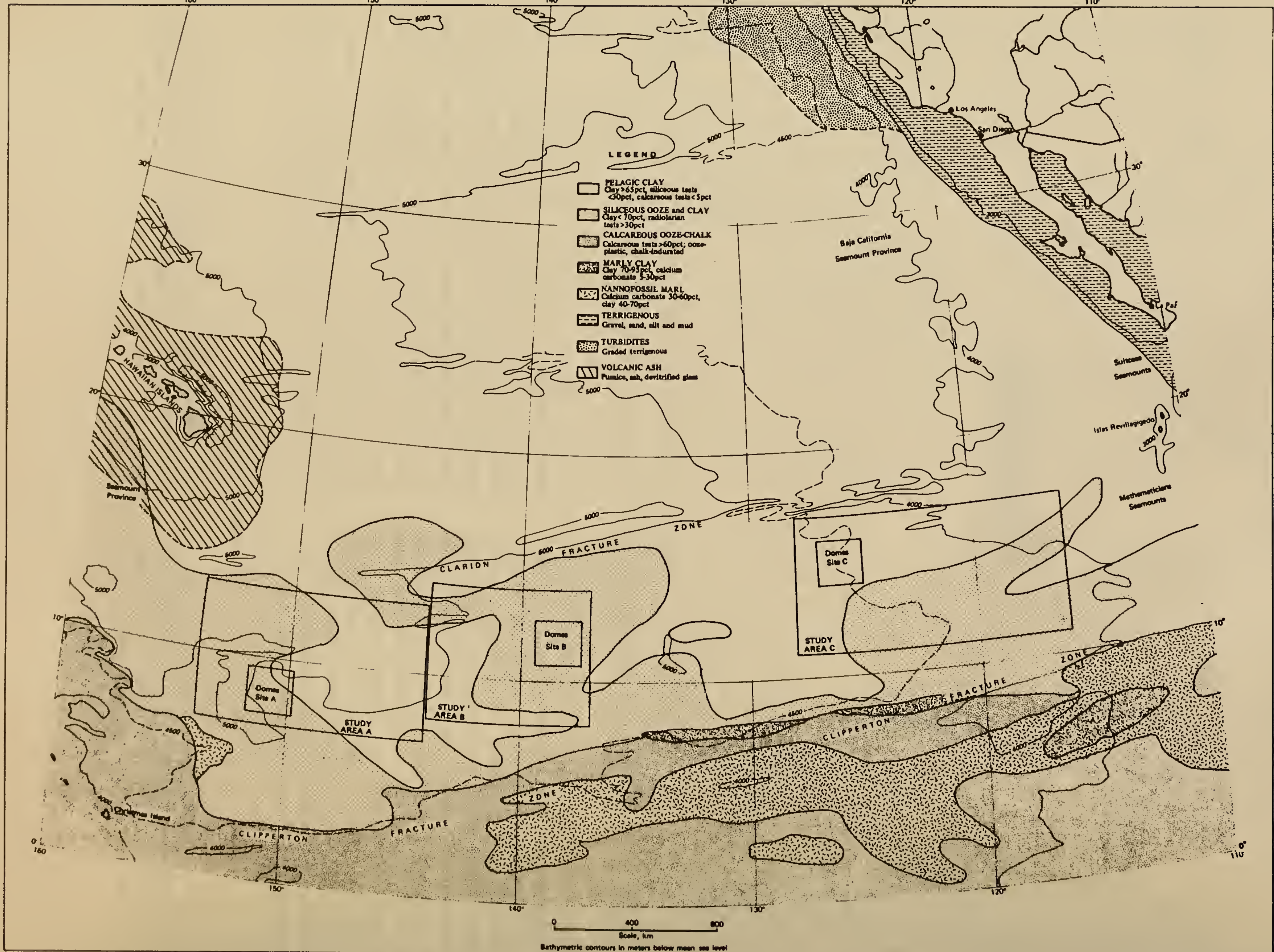

FIGURE 2. - Sediment map of the northeast Pacific Ocean, with locations of study areas A, B, and C. Modified from Horn (29) and Ryan (39). Base map AAPG (3).

LOCATION AND GEOGRAPHY

The northeast Pacific high-grade zone, presently the area of greatest commercial interest, stretches from south of Baja California at 110° W longitude to nearly 160° W longitude south of Hawaii; it encompasses approximately 10 million km^2. Study areas A, B, and C are about 4,600, 3,800, and 3,300 km, respectively, southwest of Los Angeles, CA. The best deposits generally lie between the prominent Clipperton and Clarion Fault zones.

Climate in the entire region is typical of the trade winds zone. The northeast trade winds blow steadily, but moderately, throughout the year (37). Rainfall is slight, increasing somewhat near the equator. Tropical storms and typhoons, usually lasting 1 or 2 days, occur primarily in summer months. A 10-yr survey indicates an average of 3.6, 4.4, and 2.9 storms per month for July, August, and September, respectively. The average for 1966-75 was 15 storms per year. Air temperatures at sea level average 25° C for the year; monthly averages vary only a degree or two (48).

Surface currents in the region of interest are largely controlled by wind (37). The prevailing current, the North Equatorial Current, generally flows from an east-northeast direction, splitting into several branches and eddies. Velocity measurements during the fall of 1975 and 1976 averaged 17 cm/s at the surface. The current decreases to zero at a depth of approximately 160 m and then reverses, flowing eastward at a velocity of 5.4 cm/s at 300-m depth.

Data on bottom currents in study area C are not available, yet existing water mass characteristics and theoretical studies indicate a net eastward flow at low velocities (37). Measurements at the other two study areas indicate average velocities of 2.1 cm/s in DOMES Site A, and 5.2 cm/s in DOMES Site B. A mean westward movement was recorded (37), but may be a result of the short duration of the measurement.

Assumptions made, based on the previous discussion, are that surface and bottom currents would not significantly hinder mining operations, but that tropical storms would preclude mining from 30 to 40 days each year.

GEOLOGY

GEOLOGICAL SETTING

The area of interest falls within the Eastern Pacific Sedimentary Basin. Seamounts are in all parts of the basin, but are most prevalent near the east and west margins. Groups of sediment-covered abyssal hills (33), characteristically elongated and parallel to one another, are the most dominant topographic feature. Crest-to-crest distances are variable, but typically range from 5 to 10 km (36). Local relief is generally low, but may reach 300 m. Slopes average 2° to 6° and generally do not exceed 15° except in areas of current scour or fault scarps (38). Commonly, parallel scarps impart a stairstep effect on hillsides accompanied by sediment slumping and exposure of underlying basement rock.

Water depths ranging from 3,600 to 5,500 m increase to the north and west away from the Clipperton and Clarion Fault zones and the East Pacific Rise (spreading center). Accretion of basalt crust within the narrow spreading center and progressive outward movement results in a pattern of systematic aging in a westerly direction (39). Crustal ages are mostly Oligocene in study area C, Paleocene in study area B, and Late Cretaceous in study area A.

In the northeast Pacific Ocean, distribution of sediments is strongly influenced by proximity to volcanic islands and other topographic highs, biological productivity of surface waters, and the increase of calcium carbonate solubility with depth (49).

Pelagic clays are dominant north of the Clarion fracture zone and in large irregular areas south of it (fig. 2). These sediments are reddish-brown to chocolate colored, and are composed mainly of the clay minerals illite, smectite, and kaolinite. Because dissolution of fossil remains is nearly as rapid as deposition, organic remains are generally less than 30 pct. This is attributable to a decrease in biological activity away from the equator, and an increase in water depths. Sediment accumulation rates are very low, probably from 1 to 3 mm per 1,000 yr (37).

Between the Clipperton and Clarion fracture zones, siliceous ooze and siliceous clay are the most widespread sediments. These are mostly clay minerals, but contain significant organic remains; in the case of ooze, at least 30 pct. Organic material is predominantly Quaternary radiolaria, diatoms, sponge spicules, and silicoflagellates. Accumulation rates are believed to be from 3 to 8 mm per 1,000 yr.

Calcareous sediments cover tops of isolated seamounts and the seafloor in the southwest corner of the high-grade zone, but the most widespread occurrences are south of the Clipperton fracture zone. Coccoliths and foraminifera are the chief constituents; other materials include siliceous fossils, volcanic glass, and clay minerals. Accumulation rates are variable, ranging from 10 to more than 100 mm per 1,000 yr depending on depths and surface activities (49).

Volcanic ash is the dominant sediment surrounding the Hawaiian Islands, overlapping and burying siliceous ooze and clay northwest of study area A. Terrigenous sediments, graded and ungraded, blanket the ocean floor adjacent to the North American Continent, but do not extend beyond a few hundred kilometers. Sediment accumulation rates and thicknesses in these environments are extremely variable. In the three study areas, total sediment thicknesses are thought to average as follows, in meters: area A, 250; area B, 200; and area C, 100. Wet bulk densities of sediments probably average from 1.2 to 1.3 g/cm^3 (37). Sediments higher in biogenic debris are slightly less dense, because they contain more water. Vane shear strength, which is an indicator of load-bearing strength, is variable, but may average about 20 g/cm^2 at the sediment surface. This increases rapidly to 100 g/cm^2 at a depth of 15 cm. Below 15 cm, sediment strengths are believed to remain nearly constant (37).

DEPOSIT DESCRIPTION

Nodule deposits occur mainly as irregular, single-layer fields at the sediment-water interface (fig. 3). Additional nodules buried in sediment within a meter of the ocean floor surface compose an amount approximately equal to 25 pct of those on the surface. Few nodules occur below 1-m depth (39). Within the high-grade zone, nodule sizes range from less than a millimeter (micronodules) to many centimeters in diameter. Nodule populations range from 0 to nearly 100 pct.

Highest populations and grades are usually associated with siliceous oozes and clays (19). Sedimentation rates are very low, usually a few millimeters per 1,000 yr; total sediment thickness is less than 300 m.

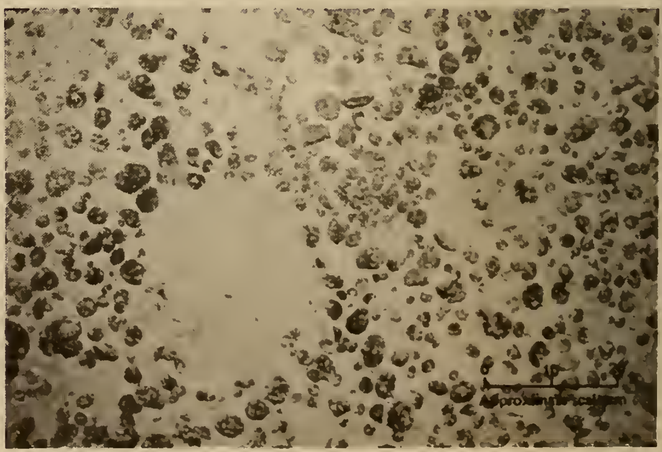

FIGURE 3. - Manganese nodule deposit in the northeast Pacific Ocean. Note irregular distribution and partial burial of nodules.

Individual nodules are dull, earthy brown to lustrous blue black, with variable shapes. Characteristically, small nodules are spheroidal, and progressively larger nodules are ellipsoidal, and finally discoidal. Sorem and Fewkes (43) attribute this phenomenon to unequal growth. Bottom portions, nested in sediment, accrete more rapidly than exposed tops. Irregular shapes, differing from the typical forms, are common. This characteristic is a result of natural agglomeration of smaller nodules, and the tendency of nodules to reflect the morphology of irregularly shaped nuclei.

Surface textures range from smooth to granular, the apparent result of different growth patterns of constituent oxides. Porosity and internal surface area of individual nodules are high, about 50 pct and 200 to 300 m^2, respectively (35). As a result, most nodules contain about 30 wt pct sea water; wet specific gravity ranges from 2.0 to 2.5

MINERALOGY

Ferromanganese nodules are typically composed of one or more nuclei surrounded by discontinuous layers of manganese and iron oxides. This imparts an onionpeel structure observed in cross section (31). Clay layers occur at irregular intervals between the oxide phases, possibly signaling periods of nongrowth. Radial and concentric fracturing are nearly universal in larger nodules (fig. 4).

According to Sorem and Fewkes (43), northeast Pacific nodules are composed mainly of dense top layers of amorphous iron oxides, whereas the bottom layers are generally intergrowths of the hydrous crystalline manganese oxides, todorokite and birnessite. Minor quantities of the mineral, δ-MnO_2 also occur. Todorokite and birnessite are believed to contain the bulk of nickel and copper present in nodules. These two metals may be carried by lattice substitution, ion exchange, or

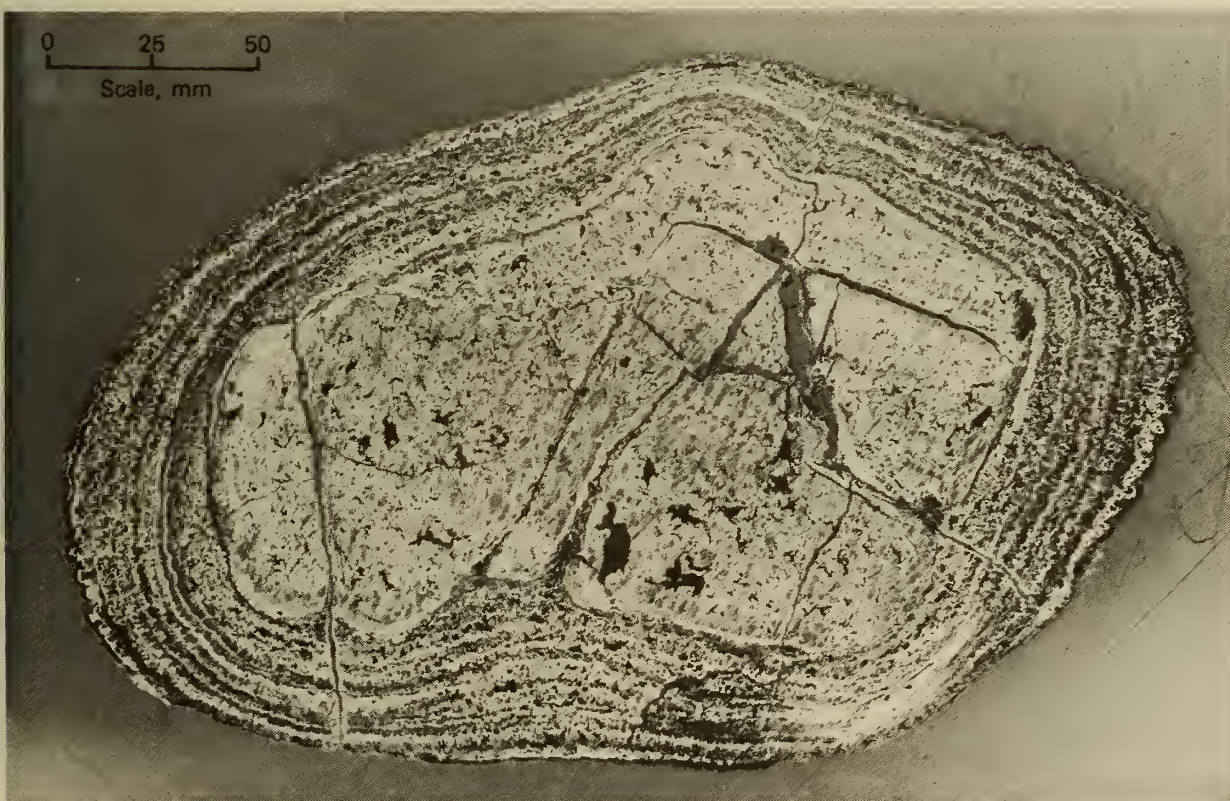

FIGURE 4. - Manganese nodule polished cross section. The large clay-rich nucleus is surrounded by concentric layers of metal-rich oxides (light) and clay (dark).

(Courtesy IFI/Plenum and Washington State University.)

adsorption (35). The cobalt association is more obscure. However, recent work indicates that in high-cobalt nodules the element is preferentially enriched in manganese oxide phases. Conversely, in nodules with lower cobalt, iron oxides contain most of the element (21).

RESOURCES

DISCUSSION

In this report, grades assigned to deposits are based on X-ray fluorescence spectrometry and atomic absorption analyses of nodules. Analytical data were obtained from many sources and have been carefully screened. As an example, assays of specific nodule parts, such as nuclei, outer layers, etc., were discarded and only analyses of whole nodules or representative portions retained.[3]

Detailed studies of samples from DOMES Site C (11) show that the arithmetic mean of metal contents at individual sample sites can be predicted within ±10 pct, and at a 90-pct confidence level, with relatively few assays. For example, at 76 pct of the sites sampled, means for nickel, copper, cobalt, manganese, iron, and zinc can be estimated from less than 20 nodule analyses per site. If only nickel and copper are of interest, the mean can be predicted at 86 pct of the sample locations with analyses of just 11 nodules per site. The study (11) also shows that variations in metal contents may be greater between nodules in the same sample than between sample averages from nearby, yet different locations. This phenomenon is probably the result of variations in metal content between different size nodules. Comparisons of hundreds of analyses by Frazer (22) support the observations of Fewkes (11-13), and indicate arithmetic means can be accurately predicted for sample stations in large areas of the high-grade zone with relatively few analyses.

Nodule abundances (weight per unit area) and consequently resource quantities calculated from them are more difficult to determine. Despite the fact that nodule deposits cover very large areas of the seafloor, local nodule populations are extremely variable. Over distances of a few meters, abundances may range from near 0 to 10 or 15 kg/m^2. Ideally, sampling is conducted on a grid basis, with bottom photography or television surveys between sample points. By necessity, however, estimates in this report are based on randomly located samples and on series of photographs taken sequentially along linear track lines. This is because the original purpose of the work was research, rather than resource evaluation.

Factors relating to abundance and resource estimates from seafloor photographs and in situ sampling are discussed in appendix A. Because detailed site-specific data are unavailable to calculate photograph correction factors and because some sampling devices may lose portions of samples, estimates in this report are probably conservative, and are considered "minimum" values. Costing and system descriptions in succeeding sections are based on these minimum values.

GRADE AND ABUNDANCE ESTIMATES

Based on available data, estimates of grades and abundances are made for deposits in study areas A, B, and C. These three large areas are divided into several smaller ones (subareas) on the basis of station locations, nodule grades, and abundances. Rectangular boundaries are used for convenience and do not necessarily enclose or delimit any single deposit. In reality, an area large enough to support commercial mining may contain several distinct deposits.

Assigned grades are simply the average of arithmetic means of samples at individual sites within subarea boundaries. Abundances are likewise averages of

[3]See Frazer (22), Fisk (14), and Frazer and Fisk (20-21) for data, references, and discussion. See Fewkes (11-13) for additional information and discussion.

abundance estimates for individual stations. Abundance estimates determined from actual samples were not differentiated from those derived from photographs, nor was any greater significance attached to them.

Station locations, average grades, and other information are illustrated in figures 5 through 10 (figs. 5, 7, and 9 are in pocket at end of report). Figures 6, 8, and 10 are enlargements of intensely sampled areas, which could not be properly shown on a small-scale map. Numbers next to sample points are keyed to corresponding index numbers in Appendix B, which is a series of tables containing index numbers, locations, population estimates, abundance estimates, nodule assays, and statistical summaries. Individual tables are included for each subarea within study areas A, B, and C. Data for samples outside subarea boundaries are listed in separate tables.

Table 1 is a summary of analytical data contained in Appendix B. Grades assigned to deposits in subareas AII, AIII, BIII, CI, and CII average 2.3 wt pct or greater nickel plus copper. This figure is regarded by some (18) to be a minimum requirement for a potential minesite. Of the remaining subareas only AIV appears to contain deposits grading well below the estimated requirement of 2.3 wt pct nickel plus copper.

A summary of available abundance data and subarea size is contained in table 2. For convenience, abundances are given in both wet kilograms per square meter and dry metric tons per square kilometer. Subareas AII, AIV, and CI appear to contain significantly high abundances, while AIII and BIII apparently contain medium abundances, and AV has low abundances. No estimates are available for the remaining five subareas.

Consideration of grade, abundance, and size (square kilometers) indicates three of the subareas (AII, BIII, and CI) have the greatest economic potential, because deposits are high grade, and occur over large areas in sufficient abundance to support mining for a reasonable length of time (i.e., 20 yr). The proposed mining and beneficiation systems and economic analyses address these three subareas only.

TABLE 1. - Summary of mean elemental contents, study areas A, B, and C, weight-percent[1]

Subarea	Ni	Cu	Co	Mo	Mn	Fe
AI.....	1.20	0.98	0.21	0.04	24.2	7.4
AII....	1.30	1.00	.21	.05	27.8	7.1
AIII...	1.30	1.19	.22	.05	26.5	7.3
AIV....	.96	.67	.28		20.2	10.0
AV.....	1.41	1.20	.21		26.2	5.7
AVI....	1.13	1.10	.17	.04	24.3	6.9
BI.....	1.10	.84	.23		22.0	6.1
BII....	1.18	.92	.23	.06	24.3	6.3
BIII...	1.45	1.24	.25	.07	27.2	5.2
CI.....	1.33	1.04	.26	.07	26.8	6.9
CII....	1.39	.96	.17	.07	27.7	8.5

[1]Dry-weight basis.

NOTE.--Blank indicates no information available.

TABLE 2. - Average abundance and subarea size, study areas A, B, and C

Subarea	Estimates	Abundance[1]		Subarea size, km^2
		Wet, kg/m^2	Dry, t/km^2	
AI.........	0	NAp	NAp	43,700
AII........	12	8.8	6,200	36,000
AIII.......	9	6.2	4,300	17,000
AIV........	16	9.2	6,400	22,100
AV.........	15	3.4	2,400	24,100
AVI........	0	NAp	NAp	18,500
BI.........	0	NAp	NAp	12,500
BII........	0	NAp	NAp	10,900
BIII.......	74	5.3	3,700	64,600
CI.........	59	11.7	8,200	57,600
CII........	0	NAp	NAp	35,800

NAp Not applicable; no estimates.
[1]Wet nodules contain approximately 30 wt pct free water.

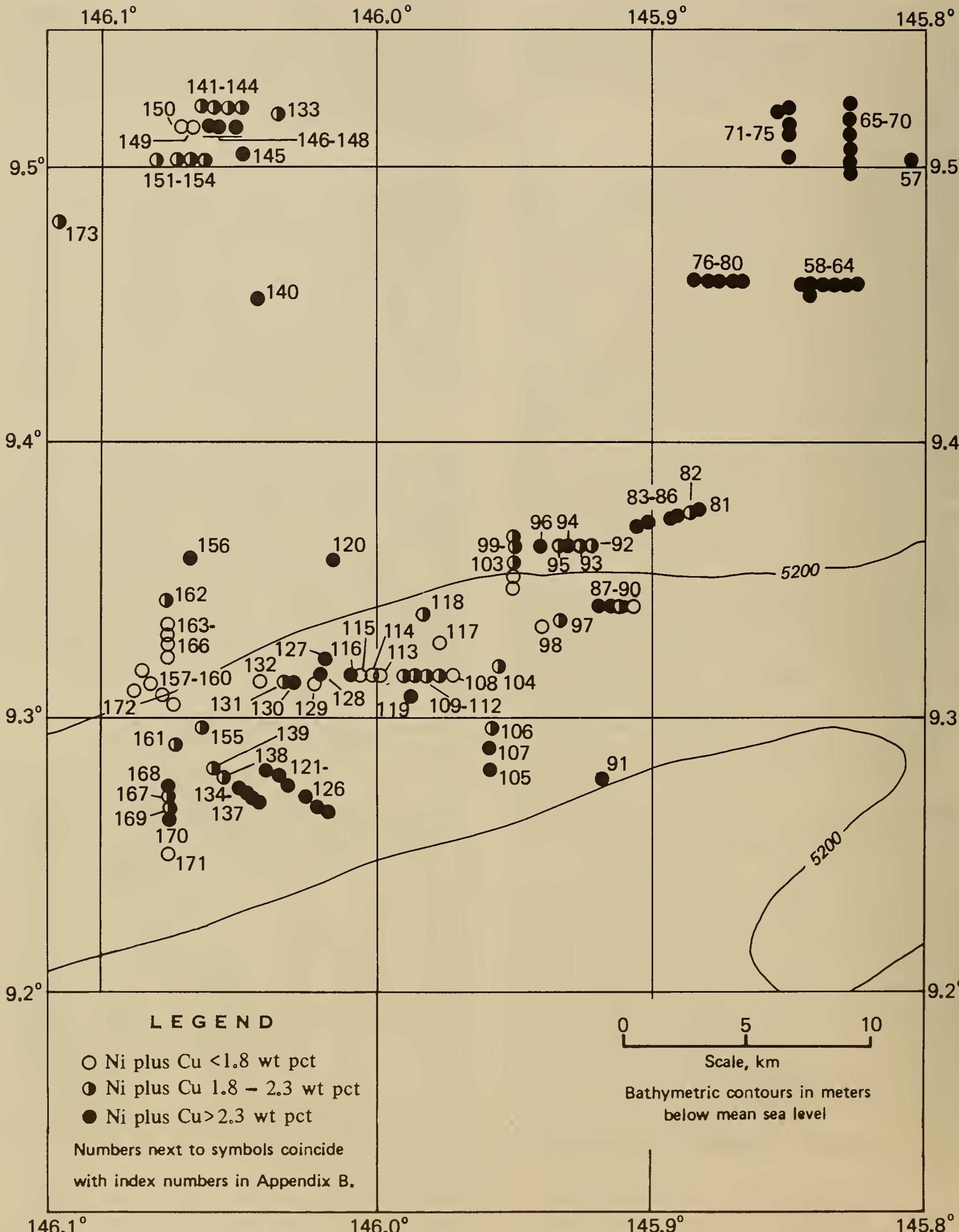

FIGURE 6. - Station locations, nodule occurrences, and grades in subarea AII(a).

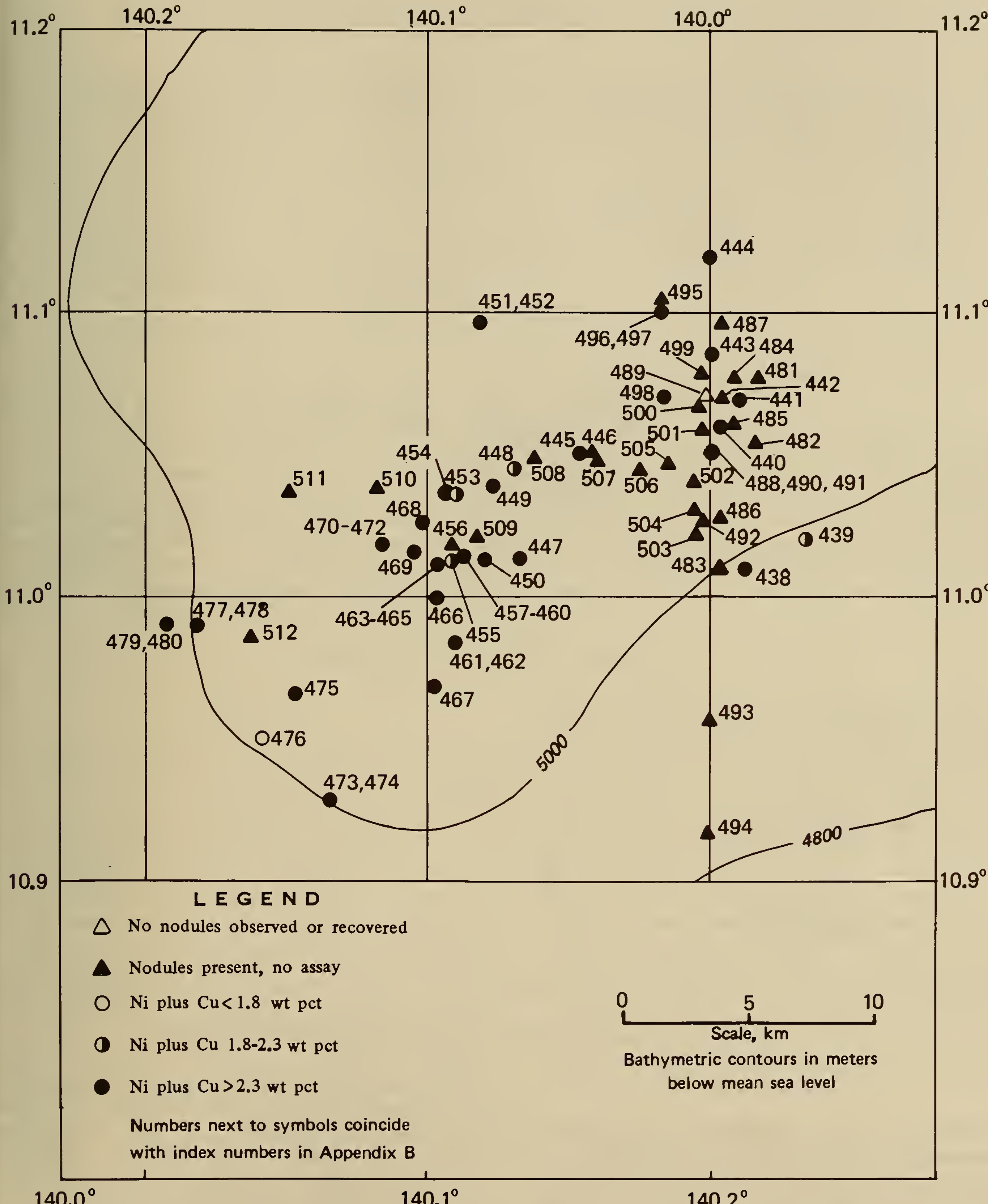

FIGURE 8. - Station locations, nodule occurrences, and grades in subarea BIII(a).

FIGURE 10. - Station locations, nodule occurrences, and grades in subarea CI.

MINABLE RESOURCES

Gross tonnage estimates for potential minesites can be very misleading. A series of practical considerations significantly lower the resource quantity that will actually be recovered. At any given site in the northeast Pacific there are fault scarps, basalt outcrops, excessive slope angles, and other seafloor features that reduce the usable portion of a deposit area to an estimated 75 pct of initial size (34). Of the remaining area, the abundance of deposits in possibly one-third of it is insufficient to warrant mining. This means that about half the original area has a minable resource. Furthermore, not all minable resource will be recovered because pickup efficiencies of presently envisioned mine

TABLE 3. - Estimated minable and recoverable resources, subareas AII, BIII, and CI

	AII	BIII	CI
Average abundance................dry t/km^2..	6,200	3,700	8,200
Minable area[1]..........................km^2..	18,000	32,300	28,800
Minable resource..............million dry t..	111.6	119.5	236.2
Minable nodules traversed..............pct..	70	70	70
Pickup efficiency......................pct..	85	80	90
Mining efficiency[2]....................pct..	60	56	63
Recoverable resources[3].......million dry t..	67.0	66.9	148.8

[1]50 pct of total minesite area.
[2]Minable nodules traversed times pickup efficiency.
[3]Minable resource times mining efficiency.

systems are expected to be no more than 80 to 90 pct. Pickup efficiency will vary because of dissimilar abundances, and other physical factors. Also, maneuvering and control limitations mean that only about 70 pct of the minable nodules will be traversed, thus mining efficiency will average about 60 pct. Considering all these factors, approximately 30 pct of the potential resource from a given deposit area will be recovered. This low percentage will probably increase with mining experience.

Based on the above analysis, table 3 contains estimates of both minable and recoverable resources for subareas AII, BIII, and CI. These estimates serve as a resource base for the proposed mining-transportation-beneficiation systems described in succeeding sections.

MINING AND PROCESSING TECHNOLOGY

DISCUSSION

Major ocean mining consortia are in various stages of designing, building, and testing manganese nodule mining and processing systems. Presently, two mining-lift systems have been tested in both experimental facilities and at sea. The first, a continuous line bucket system consists of a 10-cm-diameter polypropylene line with drag buckets attached at regular intervals. A loop is played overboard and allowed to trail on the ocean floor. Traction devices on either one or two ships provide lateral movement as the loop is towed forward. Buckets remain attached as the rope moves through the friction drives and are emptied. The obvious advantage of this system is its simplicity. However, tests by the CNEXO (French) and Sumitomo consortia in the 1970's, were not successful, apparently because of tangling and low nodule recoveries. Development of this system appears to have been terminated.

The second system is hydraulic and consists of a nodule collector unit attached to a mining ship by a steel pipeline. The pipe serves as both a towing means and a conduit for raising nodules. Lift is provided by either submersible hydraulic pumps or high-pressure air injected at a predetermined depth (air lift). Bubbles create upward movement in the pipe column as they rise and expand. One-fifth scale tests of hydraulic units were successfully conducted in 1978 by both the INCO and Deepsea Ventures consortia. Approximately 1,300 t of nodules were dredged from two sites in the northeast Pacific. Water depths were approximately 4,500 and 5,000 m.

A variety of processing schemes have been proposed for recovering value metals from nodules; the most promising have been tested in small pilot plants. Significant research has been directed towards development of four processes: (1) the Kennecott (Cuprion) process and

(2) the high-temperature sulfuric acid leach, both designed to recover primarily nickel, copper and cobalt; (3) the INCO process; and, (4) Deepsea Ventures processes, recovering manganese as well as nickel, copper, and cobalt.

The Deepsea Ventures (Ocean Mining Associates) process is based on reaction of the manganese-iron hydroxide matrix with hydrochloric acid (HCl) to produce soluble iron and manganese chlorides, and thereby releasing metal values. Ferric chloride is removed by solvent extraction and oxidized to produce recyclable HCl and iron oxide. Copper, and then nickel, cobalt, and molybdenum are separated by liquid ion exchange (LIX) from the chloride solution and electrowon in separate chloride circuits. Manganese metal is recovered by either fused salt electrolysis or reduction with aluminum metal. Alternately, manganese oxide can be recovered by high-temperature hydrolysis of $MnCl_2$.

The number and diversity of patents issued on the process suggests problems have been experienced (35). One difficulty is the large consumption of HCl (makeup requirement about 50 pct) during initial reduction and accompanying production of chlorine, which is not utilized in the remainder of the process. The gas must be marketed as a byproduct, exchanged with a polyvinyl monomer producer for excess hydrogen chloride, or reconverted to HCl. Although only minor problems occur in metal extraction and electrowinning from chloride solutions, Deepsea Ventures has not developed a satisfactory method to convert the manganese oxide intermediate product to a usable end product (ferromanganese). According to Monhemius (35), Metallurgie Hoboken-Overpelt, a new member of the consortium through its corporate ties with Union Miniere (Belgium), recently developed system modifications that may resolve some of these problems.

The INCO process is a combination of pyrometallurgical and hydrometallurgical methods that have been used in the processing of terrestrial ores. Stockpiled ore, containing about 30 pct water, is dried and then selectively reduced at 1,400° C. Two phases are produced; a manganese-rich slag and an iron-nickel-copper-cobalt alloy. The alloy is oxidized to remove most iron and manganese; and next converted to a sulfide matte by the addition of pyrite, gypsum, and coke. It is then reoxidized (blown) to remove residual iron. Slags are returned to reduction and smelting furnaces. The remaining matte is ground and pressure leached with sulfuric acid; metals are recovered by solvent extraction, electrowinning, and precipitation. Ferromanganese, with acceptable manganese to phosphorous ratios, is produced from the manganese slag by reduction smelting with lime at 1,600° C.

INCO's method has some distinct advantages. First, nearly all manganese and much of the iron is removed in a molten slag, which can be used to produce ferromanganese. Second, other valuable commodities are concentrated into an alloy phase, which weighs less than 10 pct of the original feed (35). Treatment of this smaller quantity of material is relatively cheap compared with other processes and most of the commercial technology already exists (8). The principal disadvantage of this process is the high energy requirements for drying and smelting.

The high-temperature sulfuric acid leach process, which has been investigated in European laboratories (7), is an adaptation of the technique used to recover nickel from laterites at Moa Bay, Cuba. Raw, wet nodules are ground, mixed with concentrated sulfuric acid, and heated to about 250° C. At this temperature, most of the copper, nickel, and cobalt are dissolved, while little iron or manganese enter solution; thereby, subsequent purification steps are simplified, and acid consumption is minimized. Value metals are recovered from the cooled leach effluent by essentially the same sequence of steps used in the hydrometallurgical portion of the smelting process.

Because of high temperatures and acidities in the leaching step, care must be

taken in selecting materials for construction. Another potential drawback to this process involves disposal of spent sulfate. Disposal of sulfate as gypsum, with recycling of ammonia within the process, generates large amounts of waste. An alternate approach involves the purification of ammonium sulfate for sale as fertilizer. Less information has been published on this process than with the Deepsea Ventures, INCO, and Kennecott processes.

In the Kennecott (Cuprion) process, wet ore is ground, and then slurried in a mixture of seawater and recycled process liquor which contains dissolved copper and ammoniacal ammonium carbonate. The slurry passes through a series of reaction vessels into which carbon monoxide is introduced. Cuprous ions are produced which subsequently catalyze the reduction of the manganese-iron oxide matrix (1). Value metals dissolve and are separated from the reduced residues by countercurrent washing. Ammonia and carbon dioxide are recovered and recycled by steam stripping residues. Electrowon copper and nickel are produced after having been extracted from the leach liquor using a mixture of LIX 64N[4] in kerosine. Cobalt is recovered from the remaining solution (raffinite) by precipitation with H_2S and subsequent reduction. Electrowinning is employed to recover nickel and copper as high-grade cathodes.

Several favorable factors characterize this process: nearly all process steps are carried out at ambient temperature and pressure; energy consumption is relatively low; most reagents are relatively inexpensive or recyclable; and there is only limited use of corrosive and highly toxic reagents. Apparently, these characteristics were successfully demonstrated in a 350-kg/d pilot plant, which was operated for 43 days at Kennecott's Ledgemont Laboratory (1).

[4]Reference to specific products does not imply endorsement by the Bureau of Mines.

While both the Cuprion and high-temperature sulfuric acid leach processes previously described are designed to recover primarily copper, nickel, and cobalt, it is possible that other metals such as molybdenum, would also be recovered. In this case, additional metals separation and purification steps would be required, but the process would not be greatly changed. Also, manganese could be recovered by a combination of physical and chemical steps. However, if manganese were to be recovered, materials handling and process design would be significantly altered and energy requirements would increase considerably.

OPERATIONAL INTEGRATION OF A PROPOSED RECOVERY SYSTEM

Because description and costing of a nodule mine-transport-beneficiation system is very complex, only one system is described and applied to the three subareas, AII, BIII, and CI (hereafter called ventures 1, 2, and 3). The system is modified for each subarea to reflect differences in transport distances, water depths, and nodule abundances. However, except to lower the projected mining rate for venture 2 which has comparatively low nodule abundance, no attempt has been made to optimize the many factors bearing on economic viability. Mining consortia and regulatory agencies involved would determine exact areas to be mined, their sizes, locations of facilities, methods employed, and production rates.

The proposed system is, in part, hypothetical because there has been no commercial production experience, yet the descriptions and costs are drawn from many knowledgeable sources, both published and unpublished. Because both methods were successfully tested, the system plan includes hydraulic mining-lifting and Cuprion processing. Other beneficiation methods might be as feasible, but little information is available concerning process details and test results. Slurry transfer and transport of nodule ore is considered most likely,

because nodules are amenable to the method, and much slurry handling experience exists in other areas of the minerals industry.

Mining is scheduled on a 300-d/yr basis with an estimated annual production of 3.0 million dry t for ventures 1 and 3, and 2.4 million dry t for venture 2. Two ships recovering 5,000 dry t/d each (4,000 t for venture 2) would sweep the minesite in predetermined paths. Hydraulic collectors, towed at an average velocity of 1.0 m/s (2 kn), would dislodge, sort, and channel nodules to a large-diameter pipe, which would connect the collector to the ship (fig. 11). Submersible hydraulic pumps would maintain an upward flow of water, nodules, and nodule fragments. Check-dump valves would protect the pumps, and prevent clogging in case of power failure.

Aboard the mine ship, nodules would be screened, conveyed to storage holds, and dewatered by decantation. The ore would not be upgraded because chemical and physical characteristics of nodules do not lend themselves to traditional concentration methods. Every few days, nodule ore would be reslurried and pumped through a flexible pipeline to large-capacity bulk transports where it would be again dewatered and then transported to an unloading terminal on the west coast of the United States.

At the unloading facility, ore would be reslurried and pumped to nodule storage ponds on shore. From there the slurry would be transferred inland, by slurry pipeline, to storage ponds at the processing plant.

The processing plant would operate 24 h/d, 330 d/yr at 100 pct capacity. Nickel and copper would be recovered as high-purity cathodes, and cobalt would be chemically precipitated, purified, and recovered as metallic or oxide powder. An add-on option is included to recover manganese by treating about one-half the Cuprion tailing and upgrading it to a ferromanganese product.

Tailings from both Cuprion and ferromanganese plants would be treated to adjust pH and then pumped to tailings ponds of conventional design. Slag from the ferromanganese plant would be stored in an adjacent area.

MINING

Nodule mining in each deep ocean site would be conducted 300 d/yr at a rate of 8,000 dry t/d (11,430 t/d as mined, wet) for venture 2 and 10,000 dry t/d (14,300 t/d as mined, wet) for the other two ventures. Operations would be conducted around the clock, but actual production time is estimated at 20 h/d. Hourly rates would therefore be 400 and 500 dry t, respectively. Major equipment modifications, ship repairs, and drydock would likely take place during July through August, when from three to four major storms normally occur. A high-speed boat, operating from a major port, would transport personnel and supplies to the minesite. Ship-to-ship transfer would be by helicopter.

Prior to actual mining, extensive characterization of a portion of the previously explored minesite would be completed. A large-scale bathymetric map would be constructed; locations of bottom obstructions, including cliffs, faults, and rock outcrops would be clearly marked. Sediment bearing strengths and other factors of local marine environment would be analyzed in detail. Based on this additional resource information, a mining plan would be drawn up at least a year in advance of each year of mining. The survey program would continue throughout the life of the operation, requiring the use of a survey vessel most of the time.

The mining system would be composed of two mine ships, each capable of conducting operations independent of the other. Each ship would be in the 100,000-dwt class; approximately 250 m long, with a 45-m beam and a draft of about 12.2 m. About 30,000 shaft hp would be required for propulsion, mining, ore transfer, and

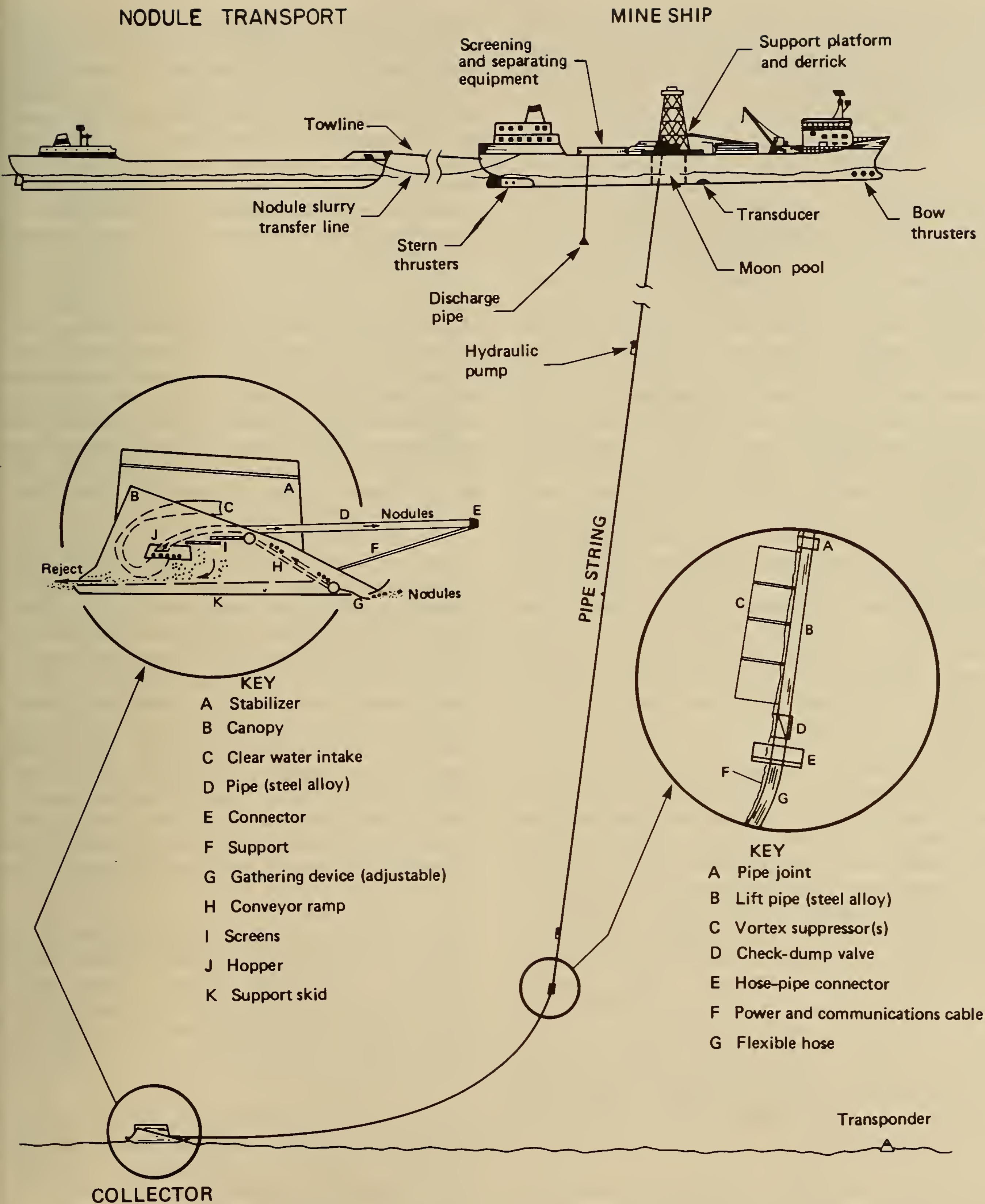

FIGURE 11. - Hydraulic deep ocean mining system. Modified from Flipse (15-16) and Grote (24).

crew accommodations (5). The large ship size is in part dictated by the need to have several days storage capacity, to significantly reduce the number of transports.

A ship of conventional hull form, similar to an ore carrier, would be used but would require many changes and additions to be able to serve as a mine ship. The most conspicuous addition, the pipe handling system, would consist of a derrick, support platform, pipe racks, and combination crane-elevator used to move pipe from storage. The derrick and platform would be built over a rectangular hole (moon pool) cut through the hull of the ship. The moon pool would be amidships where there is the least pitch and roll. Assuming pipe support at the main strength deck, a 20- by 20-m moon pool should be sufficient to keep the pipe from striking the walls during a 30° roll (17), the design maximum. The support platform-work floor, capable of holding an estimated 5,000 t, would be mounted on a two-axle gimbal, which allows the trailing pipestring to remain near vertical as the ship pitches and rolls. A hydraulic support system would compensate for heave.

Other onboard equipment would consist of separator and settling tanks, screens, and conveyors to separate nodules from sediments and other waste. Slurry pumps and permanent piping would be installed to route nodules to storage and sediments overboard; possibly through a flexible pipe extending to a maximum depth of 200 m (16). Special care would be taken to retain nodule "fines," which carry most metal values. Additional tanks and pumps would be used to reslurry the ore and transfer it to bulk transport vessels. Total mine ship storage would be 70,000 wet t (49,000 dry t), equivalent to the capacity of the ore transports plus 10 pct margin.

To tow the collector at required low velocities and to accurately maneuver, the mine ship would have to be fitted with a sophisticated computer-controlled propulsion system. This would include bow and stern thrusters as well as a sonar locator system. The locator system would consist of hull-mounted transducers that generate sound pulses. The pulses would be picked up and returned by a set of transponders positioned on the seafloor. The returned signals would be analyzed by a computer, which would steer the ship within a few feet of the prescribed course (50) and around obstacles.

A crew (mining and operational) of about 72 (17) would require living and recreational facilities. Because of extended tours of duty, these facilities might include amenities such as individual quarters, a gymnasium, theater, and gameroom. Two crews for each ship would be required to permit a 30-day-on, 30-day-off work cycle typical of the offshore oil industry.

Most of the pipeline would be similar in composition to oil-drill pipe. Inside diameter would be constant at about 40 cm, while wall thickness would vary from 1.5 to 8.5 cm. Pipe section lengths from 12 to 13 m are anticipated. Sections would be joined by clamp or threaded (tool) joints. An electrical cable would be attached to the pipe to supply power to the submersible pumps and to the nodule collector. Depending on operational experience, fairings or other types of devices might be attached to reduce drag and minimize vibration of the string (24).

Near the lower end of the pipestring, a strong, flexible hose would connect the collector and steel pipe. This would allow for undulations in topography. Total length of the pipe string would be about 15 pct greater than the depth of the area being mined.

Residual sediment and biogenic debris, water, and nodules would be moved up the pipeline using a multistage hydraulic pump system. Three pumps would be mounted along the string: one within a few hundred feet of the surface and the other two about one-third and two-thirds of the water depth. An "off-line" design

would be preferable because it would utilize a motor-pump design that eliminates the need for solids to pass through the impellers. Consequently, there would be less nodule-caused abrasion, pump wear, or chance of damage in case of power failure (17, 44).

The last major component of the mining system would be a skid-mounted collector having the simplest design possible until mining experience dictates otherwise. Front-mounted tines, cutter bar, or similar devices would dislodge nodules encountered by the collector and reject oversize material. Electric or hydraulic motor-powered conveyors would move admitted nodules, nodule fragments, and sediment through a series of screens to the hose or dredge pipe opening where hydraulic flow would move them up the pipeline. During the screening process most of the sediment would be removed.

Production demands would require a large collector, as much as 20 m in width, depending on average deposit abundance. The device would be strongly built to withstand inevitable collisions with undetected obstacles, and gross weights would probably range from 10 to 30 t.

Actual mining would begin by preparing the collector and attaching it to the flexible hose. Depending on size, the unit would be either lowered over the side and keelhauled beneath the moon pool or lowered directly through it. Steel pipe with attached fairings and power cable would be added section by section until the proper pipe string length had been achieved. Assuming 15 min per section, 4 to 5 days would be required to complete the job. The final pipe section would be connected to the separator and ore handling equipment, and mining would proceed as planned.

The collector would be towed at a velocity of approximately 1 m/s (2 kn), resulting in a pipeline trailing angle of about 7° from vertical (15). The velocity could not be increased appreciably because a 50-pct increase in speed nearly doubles the power requirements and flattens the trailing angle to approximately 14° or more. The large trailing angle increases pumping requirements and pipe abrasion. According to Flipse (15), a flow velocity of about 4.9 m/s is sufficient to lift nodules through a pipe 7° from vertical. A solids to fluid ratio of about 1:7 (14 vol pct) can be expected (17). Shaw (41) estimates that service life of a pipeline and collector would be about 12 and 6 months, respectively. Others expect twice this life (17).

Physical characteristics affecting minability of the potential minesites are summarized in table 4. Grade in each subarea is high and the resources appear sufficient to support mining at the specified rates for 20 yr or more. Variable water depths require different pipe string lengths but no significant changes in shipboard equipment design. Significant differences in collector sizes, however, are dictated by the wide range of nodule abundances. In fact, the low nodule abundance of venture 2 subarea

TABLE 4. - Deposit characteristics affecting minability, ventures 1, 2, and 3

	1--AII	2--BIII	3--CI
Depth....m..	5,200	4,800	4,600
Average abundance....dry t/km^2..	6,200	3,700	8,200
Average metal content, wt pct (dry basis):			
Ni....	1.30	1.45	1.33
Cu....	1.00	1.24	1.04
Co....	0.21	0.25	0.26
Mo....	0.05	0.07	0.07
Mn....	27.8	27.2	26.8
Recoverable resource....million dry t..	67.0	66.9	148.8

would require a very large dredge to maintain a reasonable, yet smaller production level than ventures 1 and 3. Increased size and weight of the larger collector would require a stronger and, very likely, heavier pipe resulting in increased loads on the gimballed support platform. Mine ship fuel consumption would increase, but pumping requirements would not, because the mining rate would not increase. Design of collectors for each minesite would vary in some aspects, depending on sediment type and bearing strengths, and minesite topography. However, no significant cost differences are anticipated based on design.

Table 5 summarizes mining parameters for the three ventures. Assuming 20-h days, annual production at about 95 pct capacity would be 3.0 million dry t for ventures 1 and 3, and 2.4 million dry t for venture 2.

TABLE 5. - Mining parameters for ventures 1, 2, and 3

	1	2	3
Dredge width..............m..	14	19	10
Nodules traversed[1]..dry t/h..	312	253	295
Dredge efficiency.......pct..	85	80	90
Nodules recoverd....dry t/h..	265	203	266

[1]Based on collector velocity of 1 m/s.

TRANSPORTATION

Relatively large 70,000-dwt bulk ore carriers or similar vessels are best suited to transport nodule ore because an economy of scale exists, and because they are probably the largest ships that can navigate most west coast port waters (4).[5] A modified hull design, intermediate between conventional and shallow draft types, could carry the relatively dense ore, while maintaining a reasonably shallow draft. Extra steel would be used to compartmentalize holds and give added strength (4).

Nonstructural modifications required for nodule transport vessels include piping; pumps and conveyors to receive, distribute, decant, and dewater nodule slurry; a boom to pick up and lift slurry and fuel lines aboard; and dedicated stowage tanks and piping for mine-ship fuel.

Table 6 lists dimensions and other characteristics of the proposed nodule transports. For economy, the vessels are assumed to be foreign built (European) and diesel powered. A relatively small crew of 32 would be adequate, because the ships would not be equipped to reslurry and unload their cargo. Capital costs would be less for one set of slurrying equipment at dockside, as opposed to multiple installations on transports. Also, onshore maintenance would be easier and cheaper. Shipboard slurrying equipment could be added later if, for instance, ocean dumping of tailings were to be initiated. At 90 pct of the nominal capacity of 70,000 long tons, each vessel would carry 64,000 t of dewatered slurry (4) containing the equivalent of 44,800 t of dry nodule material.

TABLE 6. - Characteristics of proposed 70,000-dwt nodule transports (4)

Dimensions, m:	
Length.........................	226.0
Beam...........................	36.9
Depth..........................	18.6
Draft........................m..	12.5
Horsepower.....................	18,700
Speed, kn:	
Laden..........................	14.5
Unladen (40 pct ballast).......	16.7
Fuel consumption, bbl/d:	
At sea.........................	490
In port........................	49
Lubricating oil...........bbl/d..	2.8

[5]Domestic ocean mining legislation (Public Law 96-283, Deep Seabed Hard Mineral Resources Act) requires processing in the United States. Alternate sites would be on the island of Hawaii and along the gulf coast.

Transport operations between the minesite and west coast would coincide with the 300-d mine ship schedule. A typical transport cycle would consist of the following four phases:

- Ship-to-ship transfer of slurry, fuel, and supplies.

- Transportation of slurry to an unloading facility.

- Unloading slurry, loading fuel and supplies.

- Returning to minesite under ballast.

Slurry transfer would be initiated by passing a towline and large-diameter, flexible pipe to the carrier. Once attached, the lines would be played out approximately 200 m (15). The mine ship would then tow the transport during pumping operations. This would insure a constant, safe distance and prevent excessive strain on the slurry line.

Normally ship-to-ship slurry pumping would be conducted simultaneously with transfer of supplies and fuel. Most supplies would be transferred by helicopter, but heavy equipment would require use of a mine-ship crane. Fuel would be pumped through a flexible line stored on the mine ship.

On the mine ship, high-pressure water jets would be sequentially directed into holds containing dewatered nodule ore. Centrifugal pumps would feed the resulting slurry through piping to a manifold on the main deck level; the slurry would then be routed to the main transfer line. The pumped slurry would consist of a maximum of 40 pct solids and have a specific gravity of 1.6 or less (5, 8). Rated capacity of individual pumps would be 500 t of solids per hour. Complete transfer would probably take 30 to 32 h, including 4 h for passing lines, connecting, playing out, and disconnecting (4).

Based on an average speed of 15.6 kn, steaming time to and from port would take an estimated 14.2, 11.5, and 9.8 days, respectively, for venture 1, 2, and 3 transports. Minor variations would be expected because of weather and sea conditions.

Unloading at an onshore slurry terminal, with a design pumping capacity greater than that of the mine ship, would require less actual pumping time than ship-to-ship transfer. However, as slurry carriers approach the facility it might be necessary to slow or wait for high tide to traverse access channels. Considering this and other delays in loading supplies and fuel, total in-port time would be 2 to 3 days.

Table 7 gives a summary of pertinent ore transportation information. The number of vessels required to meet the annual production goals of 2.4 and 3.0 million dry t, would depend on the number of trips each vessel would make per year and the capacity of each ship (44,800 dry t). In turn, the number of annual trips is the quotient of 300 operating days divided by the estimated round trip (cycle) time. Cycle time is simply the sum of loading, unloading, and steaming times.

TABLE 7. - Summary of transportation data, ventures 1, 2, and 3

	1	2	3
Distance to port [1]nmi..	2,660	2,160	1,840
Transport cycle time..............d..	18	16	13
Annual trips per vessel...............	17	19	23
Vessels required......	4	3	3

[1]nmi is equal to 1.85 km.

SLURRY TERMINAL

An onshore slurry terminal, similar to that described by Dames and Moore (8), is illustrated in figure 12. The facility, as designed, would occupy about the smallest practical land surface area and would receive, store, and pump nodule slurry to the process plant; it would not receive or load tailings for disposal at sea. A significantly larger installation would be required for that purpose,

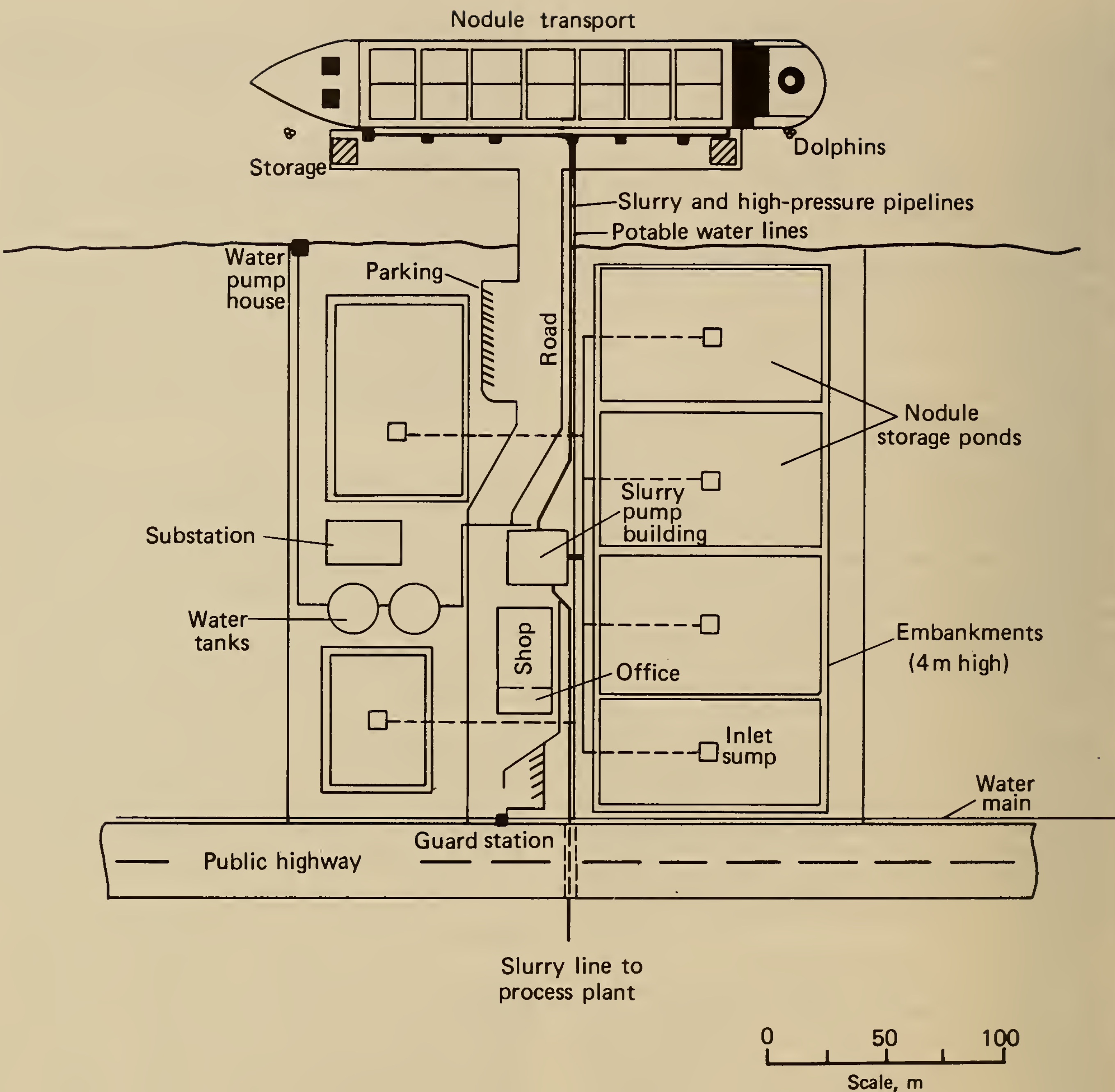

FIGURE 12. - Plan view of slurry receiving terminal and pumping station. Modified from Dames & Moore (8).

because of the large volume of waste produced by processing. Major components of the unloading facility would include a dock; mooring dolphins; 20-ton-capacity cranes to service transport vessel storage holds; a portable slurry pump for each hold; an access trestle; water and slurry piping, water tanks, raw nodule storage ponds; a pump building; and storage, shop, and office buildings.

Transport vessels from ventures 1 and 3 would arrive once every 4 to 4.5 days; vessels from venture 2 would arrive about once every 5 days. If loaded drafts were near channel depth limits, vessels would wait, then proceed to the terminal at high tide. Upon docking, hatch covers would be opened and, using dockside cranes, the portable slurry units would be suspended in each hold. High-pressure

waterlines, fed by onsite storage tanks and pumps, would be attached to slurrying nozzles. Outlets on each unit would be connected to a collector system leading to the slurry pump building and storage ponds. During unloading operations units would be lowered into the dewatered ore as special sinking jets slurried material directly below the units. Side-mounted jets would undercut and slurry surrounding ore as a hydraulic driving mechanism slowly rotated the unit. Most material would be routed to the storage ponds. However, a portion would probably be sent directly to the pump building, and then through a pipeline to the processing plant. Slurry storage capacity on the 10-ha site would be approximately 110,000 m^3, sufficient for two to three transport loads. Water requirements for reslurrying the ore would be substantial; however, most would be recycled from the process plant. Makeup water would be pumped directly from the harbor with no requirement for purification or other treatment.

Supplies for the mine ships and individual transports would be delivered to the dock by truck and lifted aboard by crane. A freshwater line would supply water when required. Diesel bunkering could be accomplished by barge, making fuel storage and pumping facilities unnecessary. Tanks and storage, however, are included in facility costs in the "Capital and Operating Costs" section.

The processing plant would probably be a significant distance inland because of limited availability of suitable land near developed ports. Two pipes would be laid side by side the entire distance. A larger pipe would carry nodule slurry to the plant, while a smaller one would return decanted water to the terminal. Assuming a maximum distance of 40 km and relatively flat terrain, a bank of centrifugal pumps with two booster stations would supply sufficient pumping capacity. Storage ponds at the terminal would allow minor pipeline repairs during the operating year; major replacements could be made in the off-season. The slurry line would be sized to transport nodules at essentially the same rate as received: approximately 425 and 340 t/h dry solids for the 3.0 and 2.4 million t/yr operations, respectively. A 25-cm-ID pipe would be required for the former operations and a 20-cm-ID pipe for the latter. Slurry densities would be somewhat lower than for other transport operations, around 30 pct solids, to allow for easy resuspension in the event of pump failure. If desirable, nodule ore could be ground at the terminal; this would reduce flow velocity and pipe diameter requirements. Operational downtime would be about 10 pct. Land usage would be about 1.0 ha for the two pumping stations and 58 ha for the pipeline. Power would be purchased from a local supplier.

CUPRION PROCESSING

The 3 million dry t processing plant would require approximately 90 ha of land. Requirements for the smaller 2.4 million dry t plant would not be much less. Both Cuprion plants would use significant amounts of water, which is assumed to be available from nearby sources. Maximum reuse of water would be practiced in the plant, and generation of electricity in the plant's utilities section would meet most power requirements. Coal would be the cheapest fuel and source of carbon monoxide reducing gas. A well-developed local infrastructure is assumed to exist nearby to support plant operations, including a pool of skilled labor, and other community services such as schools, hospitals, and fire protection. Provision is made for construction of 8.0 km of paved highway and rail spur line.

An assumption is made that waste will be stored a significant distance from processing facilities because of continuing environmental concern over waste disposal. Costs are based on a plant-to-storage distance of 100 km. Approximately 480 and 600 ha of land would be needed at the disposal site for a 20-yr operation for the smaller and and larger plants, respectively.

During normal operations, slurried ore would be received from the pipeline and be routed to storage ponds to await processing. Ponds, 5 m deep covering 12 ha, would be sufficient for 3.0 to 3.5 months' storage, a requirement dictated by the inability to mine nodules all year round. The equivalent of 9,090 dry t/d for ventures 1 and 3 or 7,270 dry t/d for venture 2 would be processed 330 d/y. Slurry would be reclaimed from storage and transferred to a surge tank in the ore preparation section, mixed with recycled ammonia-rich liquor from the reduction and leach-wash sections (fig. 13), and then fed to hydrocyclones for classification. Underflow would be fed to ball mills where particles would be reduced in size and sent back to the surge tank. Minus 100-mesh overflow, containing less than 5 pct solids (35), would report directly to the reduction section.

Ninety-eight percent of the manganese oxide matrix would be reduced in a cascade series of reaction vessels by the action of carbon monoxide catalyzed by internally generated cuprous ions.

The net effect would be to reduce manganese to the divalent state and produce manganese carbonate; as a result, metallic commodities to be recovered would be liberated. The required concentration of cuprous ions would be maintained by continually sparging a carbon monoxide-rich gas into reaction vessels. Synthesis gas generated from gasified coal contains a significant amount of H_2 and acid gases (CO_2, H_2S), which would be separated from carbon monoxide prior to its use in reduction (7).

The critical reactions were demonstrated at ambient pressure and 50° C, but a holdup time of about 20 min for each of the five vessels would be required (1). Autoclaves, maintained at a pressure of about 5 kg/cm^2 and 50° C may be utilized, thereby improving reaction rates and reducing the size of reactor vessels (35). Exothermic heat of reaction would be removed in heat exchangers and excess gas sent to ammonia recovery.

Dilute slurry would be passed to a clarifier, whose overflow would be treated with ammonia and carbon dioxide, cooled, and returned to leach reactors. Thickened underflow would be combined with second stage wash liquor and oxidized with air to convert cuprous copper to the cupric state to facilitate metal extraction by liquid ion exchange (LIX). Also, cobalt and iron would be oxidized; iron would precipite as an insoluble hydroxide (8). Offgases would be vented to ammonia recovery.

Oxidized slurry would be pumped to a countercurrent decantation (CCD) system where metals would be further solubilized, and where primary liquid-solid separation and washing would be made. Nickel and copper recovery would be greater than 90 pct (1), and cobalt recovery about 65 pct.

Manganese-rich tailings and process gas would be sent to ammonia recovery, and pregnant liquor to the LIX section for metal separation. There, nickel and copper would be coextracted, then selectively stripped (Kennecott researchers determined that coextraction of the two metals greatly simplified the process). Extraction would be carried out at 40° C in a series of three mixer-settler tanks using 40 pct LIX 64N in kerosine. Recovery of nickel and copper would be greater than 99.9 pct, but about 5 pct of the ammonia would also be extracted (2) and, if not removed, would accumulate in the nickel electrolyte. Two wash sections would reduce it to an acceptable concentration and the recovered ammonia would be recycled to the plant ammonia recovery section.

A small amount of cobalt would be extracted with nickel and copper and would have to be removed to prevent excess buildup. To accomplish this, hydrogen sulfide would be introduced into a solvent purge stream (8). The resulting precipitate would be filtered, washed, and passed to cobalt recovery.

Nickel would be selectively stripped from the organic liquor by contact with

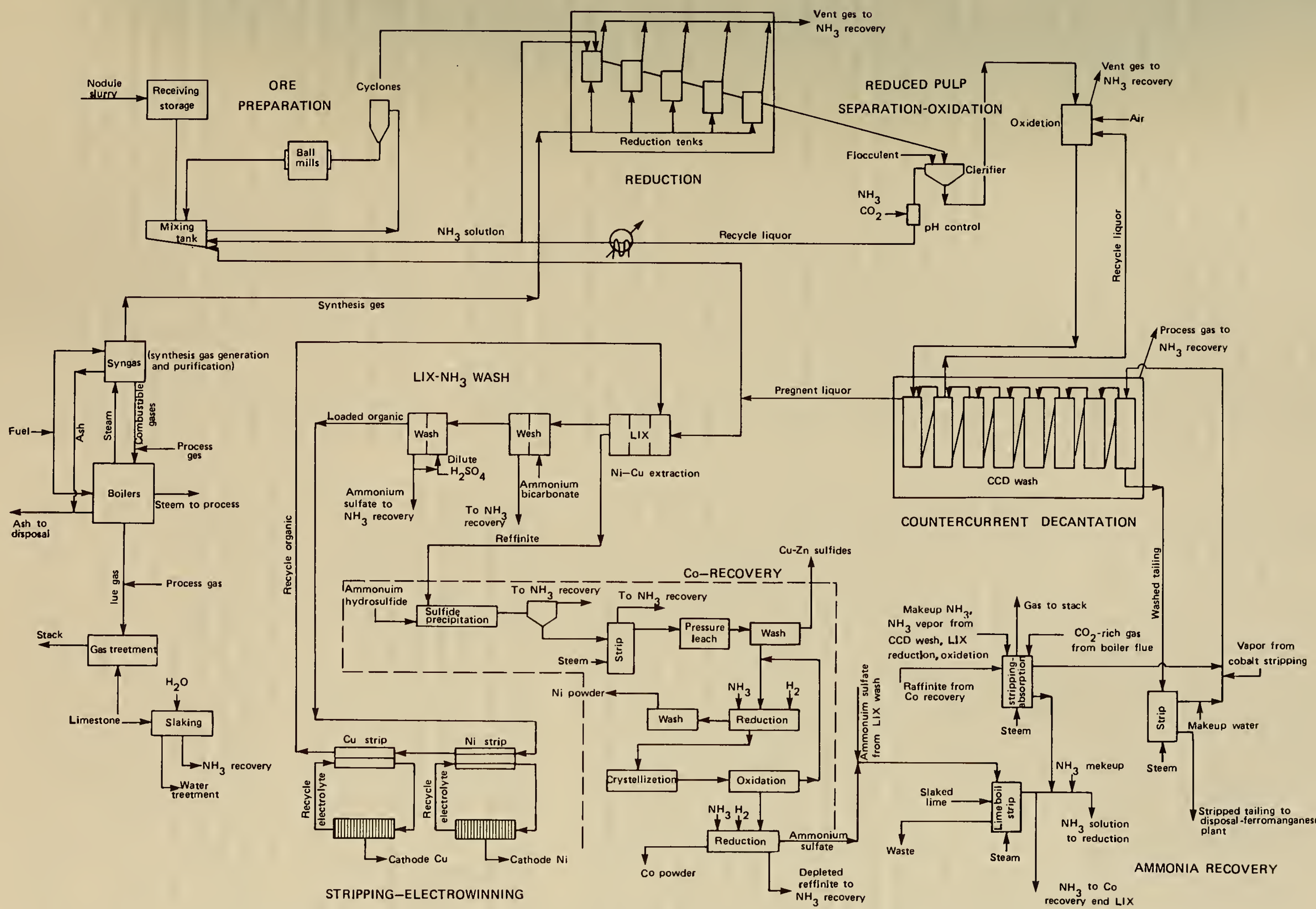

FIGURE 13. - Cuprion process flowsheet for recovery of nickel, copper, and cobalt.

acidic return electrolyte from nickel electrowinning. Composition of the electrolyte would be controlled to maintain the required selectivity for nickel; advance electrolyte, reporting to electrowinning, would contain approximately 75-g/L nickel at a pH of 3.0 and a nickel-copper ratio of 25,000:1. Stripping would take place in three stages of mixer-settlers of conventional design.

Organic feed passing to copper stripping would contain copper and nickel in a ratio of about 70:1. Both metals would be stripped, in two stages, using a more acidic return electrolyte from copper electrowinning. Nickel content of the electrolyte does not affect copper electrowinning if concentrations are below 20 g/L. This would be accomplished by bleeding copper stripping tanks. Bleed solution would be passed through a series of cells, where copper would be electrowon to depletion, and the remaining electrolyte passed to vacuum evaporators, where water would be removed and nickel sulfate precipitated. The sulfate would be sent to cobalt recovery and the remaining, strongly acidic solution returned to process, where it would be utilized to redissolve scrap copper (8).

Nickel would be electrowon from the advance electrolyte in conventional cells. Cathode bags would be used as blanks for starter sheets, and boric acid and sodium sulfate would be added to regulate pH and conductivity. Starter sheets would be cleaned in sulfuric acid prior to use. Nickel scrap would be recovered by dissolution in ammonia-rich raffinite and routing to stripping (8).

Copper would also be recovered using conventional electrowinning technology. Starter sheets would be deposited on titanium blanks in the stripping section and then installed in commercial cells. Most of the spent electrolyte would be recycled to stripping, and the remainder purged of nickel in the manner described above. Full-sized nickel and copper cathodes are to be the final products. Both commodities would be removed from cells, washed, and prepared for shipment to market.

Apparently, recovery of cobalt from the nickel-copper poor LIX raffinite has not been fully tested at pilot-plant scale. However, several plausible schemes exist. One method, outlined in the Dames and Moore (8) report is described here. Unextracted nickel, copper, cobalt, and zinc are precipitated with ammonium hydrosulfide, produced by sparging hydrogen sulfide into ammonia solution. Precipitate and solution are separated in a clarifier; overflow reports to ammonia recovery and underflow is sent to steam stripping for removal of ammonia. Sulfide slurry is mixed with purges from electrowinning, and pressure leached with air to preferentially dissolve and remove cobalt and nickel. Copper and zinc sulfide precipitates are separated, washed, and sold to smelters as concentrates.

Following pH adjustment, the nickel-cobalt sulfate solution passes to a heated autoclave where hydrogen gas is sparged into the vessel, thereby reducing the nickel. Ammonia is added to neutralize the sulfuric acid being formed. The process is completed in a series of cycles, with care being taken to prevent overreduction resulting in coprecipitation of cobalt. Once adequate density is obtained, the nickel powder is removed, washed, dried, and briquetted for sale.

The remaining sulfate solution passes to an evaporator where cobalt and small amounts of nickel and ammonium sulfates are precipitated. Nickel and cobalt salts are redissolved in strong ammonia solution and cobalt is reoxidized with air to the cobaltic state. Cobalt then remains in solution as the stream is acidified; nickel salts are precipitated and recycled to the pH adjustment step. Nickel-free solution is heated and reduced with hydrogen in an autoclave and enough ammonia is added to neutralize any acid formed. Cobalt metal is removed, washed, dried, and briquetted for sale as cobalt powder.

Sulfate purge from cobalt recovery is combined with additional ammonium sulfate from the LIX section, reacted with slaked lime, and stripped of ammonia by introduction of steam. Gypsum formed in this process is combined with other solid and liquid wastes and plant runoff. The material is treated as required and pumped to waste impoundment. Gypsum and other wastes may be combined with part or all of the manganese tailing depending on whether manganese is recovered for sale.

Manganese carbonate tailings from the CCD wash report to ammonia recovery where they are heated and stripped of ammonia and carbon dioxide by countercurrent contact with steam. Gases are combined with vapors from other ammonia stripping operations and are cooled and condensed. Scrubbed gases from process vents are also added and the solution recycled to CCD wash. Carbon dioxide makeup is obtained from boiler flues and ammonia-free gases from process vents are sent to the main stack.

Concentrated ammonia solution, needed for reduction of the raw nodule slurry, is obtained by partially stripping raffinite from cobalt recovery; the resulting ammonia-rich vapors are combined with vapors from LIX, reduction, oxidation, and lime boil steps as well as makeup ammonia. The gas is condensed and recycled to nodule reduction.

Table 8 lists material, water, and power requirements of a 3 million dry t/yr Cuprion plant. The table also shows wastes produced, but it does not consider additional requirements for ferromanganese production. Quantities for a 2.4 million t/yr operation would be approximately 80 pct of these values.

MANGANESE RECOVERY[6]

A schematic description of an optional onsite facility to further treat part of the manganese carbonate residue (4,250 t/d) is presented in figure 14. Not all tailing is processed, because of presumed market limitations. Plant design, capacity, and fixed capital costs for all three ventures are assumed identical. Carbonate tailing from ammonia recovery would be pumped to the beneficiation section where it would be washed and centrifuged. Thickened pulp would be mixed with fresh water, soda ash, caustic soda, and sodium silicate prior to flotation with saponified fatty acids. As much as 60 pct of the manganese carbonate could be recovered in the froth as a concentrate assaying 35 to 40 pct manganese. After thickening to about 50 pct solids, the material would be dried and calcined in a rotary kiln to make synthetic manganese oxide assaying between 55 and 60 pct manganese.

When cooled, calcine would be conveyed to a surge pile or directly to one of

[6]G. D. Gale, metallurgist, Western Field Operations Center, provided design and cost information for recovery of ferromanganese.

TABLE 8. - Cuprion ammonia leach plant, major inputs and wastes[1]

Coal................thousand t..	690	Tailings.............million t..	3.3
Water...............million m^3..	8.5	Liquids.............million m^3..	4.5
Power[2]............million kW·h..	434	Gases..........thousand m^3/min..	10.3
Materials and supplies[3]..........thousand t..	270	Manufacturing.......thousand t..	125

[1]Quantities are adjusted from reference 8.
[2]Gross requirements, mostly generated internally.
[3]All supplies including liquids and gases.

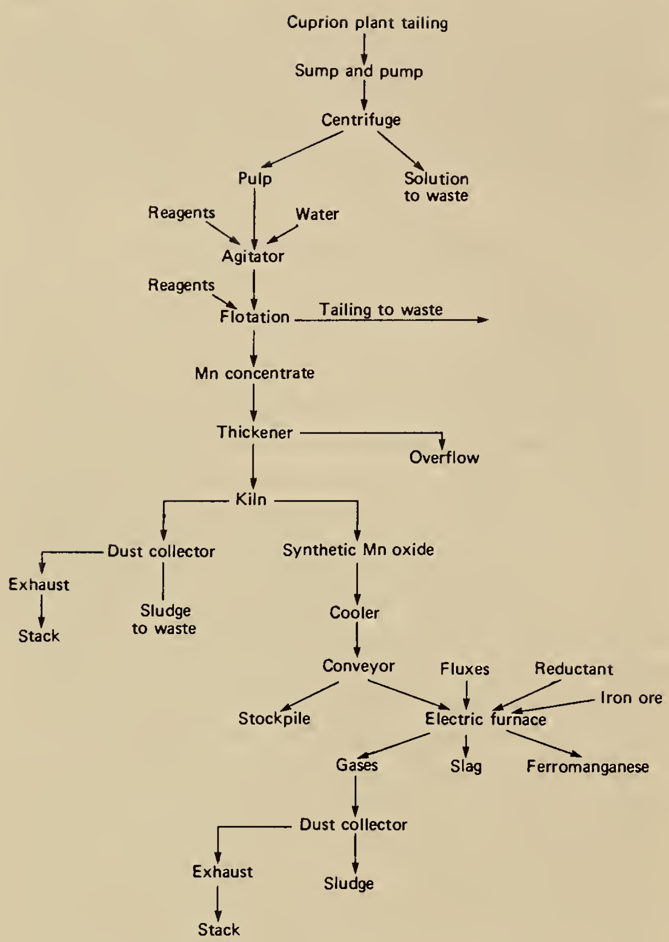

FIGURE 14. - Generalized flowsheet of a proposed ferromanganese plant.

three 25-MW, submerged resistance furnaces. The furnaces would be charged with calcine as well as limestone, silica flux, and iron ore; coke would be added as a reductant. Daily materials consumption at full capacity would be about 1,000 t manganese oxide calcine, 320 t limestone, 90 t silica flux, 90 t iron ore, and 320 t coke. Slag would be skimmed, granulated, and combined with other wastes for disposal. Molten ferromanganese at approximately 1,400° C would be poured in molds, cooled, and prepared for shipment.

Liquid wastes from the washing and concentration steps would be combined with tailing, treated to adjust pH, and pumped to tailing disposal. Dust and sludge would be recycled to the maximum extent possible, and after suitable treatment, purge streams would be disposed of with other manufacturing wastes.

Assuming the existence of an adequate market, the ferromanganese plant would operate 330 d/y, processing about 1.4 million t of tailing and producing approximately 210,000 t of ferromanganese containing 78 pct manganese. Facilities would require approximately 30 ha of land. Power would be purchased.

WASTE DISPOSAL

Tailings from the Cuprion plant and ferromanganese operation, would be pumped to conventional tailings impoundments which would be as much as 100 km from the process plant. Manufacturing wastes, such as lime boil and stack gas scrubber sludge, combustion ash, and sludge derived from water treatment operations would be treated by well-established techniques and combined with tailings for disposal.

Tailing impoundments probably would be totally enclosed and lined with clay or other impermeable material. Granulated slag could be a secondary source of bank material if a ferromanganese plant were in operation. As active ponds fill, new ones would be constructed alongside. In accordance with county, State, and Federal regulations, inactive ponds would be covered with topsoil or treated in various ways to stabilize them. A decanting system would remove and recycle water to the process plant.

CAPITAL AND OPERATING COSTS

DISCUSSION

Capital and operating costs are based on equipment, capacities, and operating parameters discussed in the previous section of this report. Costs were obtained from many sources, some unpublished. Those sources most relied upon include Flipse (17), Brown (7), and A. D. Little (6), mining and beneficiation

costs; Andrews (4-5), transportation and ship costs; Shaw (41) and Tinsley (45), general costs. Gale (23) provided ferromanganese plant costs. All figures are in January 1981 dollars. Indexing data were supplied by Flipse (17) and a Bureau of Mines cost index computer program.

CAPITAL COSTS

Tables 9, 10, and 11 contain capital cost information on mining, transportation, and processing for each of the three potential mining ventures.

TABLE 9. - Mine capital costs, ventures 1, 2, and 3, million 1981 dollars

	1--3.0 million dry t/yr	2--2.4 million dry t/yr	3--3.0 million dry t/yr
Exploration (6 yr)	$19.8	$19.8	$19.8
Research and development	69.3	69.3	69.3
2 mine ships	193.8	198.0	189.5
3 pipelines (1 spare)	55.1	66.1	50.1
2 collectors	7.3	10.1	6.6
2 pumping systems	33.0	30.6	26.0
2 sets of onboard equipment	76.2	74.8	76.2
Total fixed capital	454.5	468.7	437.5
Startup costs[1]	21.2	21.2	21.2
Working capital (1.5 yr)[2]	63.0	70.0	62.7
Total mine investment	538.7	559.9	521.4

[1]Startup capital covers extraordinary costs associated with nodule collector testing and debugging.

[2]Based on operating costs about 20 pct greater than normal. The higher unit operating costs result from lower startup capacity. During the 1st 1.5 yr an estimated 2.25 million t would be mined by ventures 1 and 3, 1.8 million t would be mined by venture 2.

TABLE 10. - Transportation capital costs, ventures 1, 2, and 3, million 1981 dollars

	1--3.0 million dry t/yr	2--2.4 million dry t/yr	3--3.0 million dry t/yr
Transport vessels[1]	$256.1	$192.1	$192.1
Slurry terminal	36.0	29.5	36.0
Slurry pipeline	18.2	16.0	18.2
High-speed supply boat	1.4	1.4	1.4
Total fixed capital	311.7	238.0	247.7
Working capital (1.5 yr)[2]	38.3	28.7	30.5
Total transportation investment	350.0	266.7	278.2

[1]4 transports for venture 1, 3 transports for ventures 2 and 3.

[2]Based on operating costs about 20 pct greater than estimated unit costs during full production. The higher costs result from lower startup capacity. During the 1st 1.5 yr approximately 2.25 million t would be transported by ventures 1 and 3, about 1.8 million t would be transported by venture 2.

TABLE 11. - Cuprion and ferromanganese plant capital costs, ventures 1, 2, and 3, million 1981 dollars

	1--3.0 million dry t/yr	2--2.4 million dry t/yr	3--3.0 million dry t/yr
CUPRION PLANT			
Research and development	$72.7	$72.7	$72.7
Plant and equipment	387.0	331.0	387.0
Utilities and services	166.2	142.2	166.2
Waste disposal	26.4	22.6	26.4
Pipeline to waste disposal	18.4	15.1	18.4
Land	2.4	2.2	2.4
Railroad spur and access road	5.1	5.1	5.1
Total fixed capital	678.2	590.9	678.2
Working capital (1.25 yr)[1]	78.8	65.7	78.8
Total Cuprion investment	757.0	656.6	757.0
FERROMANGANESE PLANT			
Plant and equipment	74.8	74.8	74.8
Utilities and services	18.0	18.0	18.0
Total fixed capital	92.8	92.8	92.8
Working capital (1.25 yr)[1]	36.5	35.2	36.5
Total ferromanganese investment	129.3	128.0	129.3
Total investment	886.3	784.6	886.3

[1]Based on operating costs averaging about 1/3 more than full production costs. During the 1.25-yr startup period, ventures 1 and 3 would process an estimated 1.63 million t of nodules, venture 2 would handle 1.30 million t. Tailing processed to ferromanganese would be about 850,000 t for ventures 1 and 3, 650,000 t for venture 2.

Included in mine capital are monies for 6 yr of prospecting and detailed mine exploration. Once mining commences, this expense would be treated as an operating cost. Research and development capital of $142 million is split almost evenly between mine and processing (17). Working capital is based on 1.5 yr operating expenses for mine and transportation, and 1.25 yr for processing. Additional capital is allowed for mine startup because of the extraordinary cost of testing and debugging the collector. Mining ships are assumed to be new and constructed in U.S. shipyards. Dredges, pipelines, and onboard slurry handling equipment are also assumed to be built in the United States. Transportation capital includes purchase of European-built transport vessels, construction of a slurry terminal and pipeline to the plant, and initial expenses of a high-speed supply boat.

Capital for the Cuprion and ferromanganese plants does not include expenses for infrastructure, such as a townsite, but does include money for a rail spur line and road to the property; a 100-km slurry line to waste disposal; all land purchases; and installation of turbines for internal generation of power sufficient for most Cuprion process requirement needs. Power for the ferromanganese plant would be purchased.

The most significant differences in capital requirements among ventures 1, 2, and 3 involving mining and transportation are increased costs associated with the larger collector for venture 2 and the purchase of an additional transport for venture 1. However, the largest single factor bearing on capital costs is Cuprion processing capacity difference between venture 2 and the other two operations. Total capital requirement is estimated to be about $100 million less for venture 2.

OPERATING COSTS

Tables 12, 13, and 14 contain estimates of operating costs for mining, transporting, and processing nodule ore. Maintenance and repair and labor costs are by far the largest expenses, accounting for 64 pct of total operating expenses. Operation of the terminal-to-plant pipeline, fuel, and fixed vessel expenses compose the bulk of transportation costs.

The bulk of Cuprion processing costs are attributable to miscellaneous capital charges, fuel (coal), and utilities and labor. Electrical power is by far the largest single cost item for operation of the ferromanganese plant.

TABLE 12. - Mine operating costs,[1] ventures 1, 2, and 3, million 1981 dollars

	1--3.0 million dry t/yr	2--2.4 million dry t/yr	3--3.0 million dry t/yr
Wages and benefits	$18.4	$18.4	$18.4
Subsistence and supplies	2.0	2.0	2.0
Maintenance and repair	27.2	30.8	26.6
Insurance	5.5	5.7	5.3
Fuel	8.6	9.6	8.0
Lubrication and oil	.1	.2	.1
Exploration	6.0	6.0	6.0
Total direct costs	67.8	72.7	66.4
General and administrative	3.7	4.0	3.7
Total annual costs	71.5	76.7	70.1
Cost per dry metric ton	$23.80	$32.00	$23.40

[1]Based on operation of 2 complete systems.

TABLE 13. - Transportation operating costs, ventures 1, 2, and 3, million 1981 dollars

	1--3.0 million dry t/yr	2--2.4 million dry t/yr	3--3.0 million dry t/yr
Wages, benefits, subsistence, maintenance and repair, insurance	$15.7	$11.8	$11.8
Fuel	12.8	8.7	9.0
Port charges	1.2	.9	.9
Helicopter crew (lease)	.8	.6	.6
Total transport costs	30.5	22.0	22.3
Supply vessel (including moorage)	1.7	1.7	1.7
Total vessel costs	32.2	23.7	24.0
Unloading and storage	2.7	2.2	2.7
Slurry pipeline (to plant)	5.6	4.6	5.6
Total direct costs	40.5	30.5	32.3
General and administrative	2.4	1.7	2.0
Total annual costs	42.9	32.2	34.3
Cost per dry metric ton	$14.30	$13.40	$11.40

TABLE 14. - Cuprion and ferromanganese plant operating cost, ventures 1, 2, and 3, million 1981 dollars

	1--3.0 million dry t/yr	2--2.4 million dry t/yr	3--3.0 million dry t/yr
CUPRION PLANT			
Materials and supplies	$4.0	$3.3	$4.0
Utilities and fuel (coal)	35.9	29.4	35.9
Labor	16.5	13.5	16.5
Waste disposal	6.9	5.6	6.9
Capital charges (maintenance, materials and supplies, insurance)	37.0	31.7	37.0
Pipeline to waste disposal	2.4	2.1	2.4
Railroad spur line	.2	.2	.2
Total direct costs	102.9	85.8	102.9
General and administrative	5.1	4.3	5.1
Total annual plant costs	108.0	90.1	108.0
FERROMANGANESE PLANT[1]			
Materials and supplies (including maintenance)	22.2	22.2	22.2
Utilities and fuel	48.9	48.9	48.9
Labor	14.4	14.4	14.4
Capital charges	6.3	6.3	6.3
Total direct costs	91.8	91.8	91.8
General and administrative	4.6	4.6	4.6
Total annual plant costs	96.4	96.4	96.4
Total annual cost (both plants)	204.4	186.5	204.4
Cost per dry metric ton:			
Cuprion plant	$36.00	$37.50	$36.00
Ferromanganese plant[2]	32.10	40.20	32.10
Both plants	68.10	77.70	68.10

[1]Includes flotation, calcining, smelting, and handling of the final ferromanganese product. Capacity would be 1.4 million t/yr for all ventures.

[2]Calculated on basis of total dry metric tons of nodule material processed, not just quantity processed in ferromanganese plant.

Operating cost differences among the three ventures somewhat parallel differences in investment costs. The larger, heavier collector is primarily responsible for high costs of venture 2 mining, while running an extra transport vessel over longer distances makes transportation of venture 1 ore most costly. Economy of scale for Cuprion processing results in slightly cheaper processing of nodules for ventures 1 and 3.

COST SUMMARY

Capital and operating costs discussed in previous sections are summarized in tables 15 and 16. Initial investments are large, ranging from about $1.5 to nearly $1.7 billion for a three-metal nickel, copper, and cobalt operation. A plant to process approximately 1.4 million t tails annually to produce ferromanganese would require about $130 million additional capital. Estimates of operating costs are from $71 to $83 per dry metric ton without manganese recovery and from $103 to $123 per dry metric ton with production of ferromanganese.

Probably no effective means exist to significantly reduce capital costs for

TABLE 15. - Capital cost summary, ventures 1, 2, and 3, million 1981 dollars

	1	2	3
Mining	$538.7	$559.9	$521.4
Transportation and transfer	350.0	266.7	278.2
Cuprion processing	757.0	656.6	757.0
Total	1,645.7	1,483.2	1,556.6
Ferromanganese plant[1]	129.3	128.0	129.3
Grand total	1,775.0	1,611.2	1,685.9

[1]Capacity, and consequently investment, would remain constant for the ferromanganese plants except for slightly less working capital for venture 2; a result of lower startup capacity.

TABLE 16. - Operating cost summary, ventures 1, 2, and 3, 1981 dollars per dry metric ton

	1	2	3
Mining	$23.80	$32.00	$23.40
Transportation and transfer	14.30	13.40	11.40
Cuprion processing	36.00	37.50	36.00
Total	74.10	82.90	70.80
Ferromanganese plant	32.10	40.20	32.10
Grand total	106.20	123.10	102.90

mine, transportation, or beneficiation systems as conceived and described. However, operating costs, particularly mine operating costs, might decrease as experience is gained. Other mine systems such as a continuous line bucket, could possibly cost less to build and operate, but technical feasibility has yet to be demonstrated. Scaling down to 1 million dry t/yr, for example, would reduce capital, but increase unit operating costs.

A reduction in transportation costs could be realized for ventures 1 and 2 by placing the processing plant nearer the minesite, possibly on the island of Hawaii. The number of transports and distances traveled would be reduced but increased energy and land costs could possibly offset savings.

A savings of plant capital could be accomplished by purchasing power for operations instead of installing generators or by using oil-burning rather than coal-burning equipment. Both actions, however, would probably result in significant increases in operating costs.

FINANCIAL ANALYSIS[7]

Financial analyses were carried out using the Bureau of Mines MINSIM4 computer program (47). This program calculates either the discounted cash flow rate of return (DCFROR) or determines a product value requisite to achieve a given DCFROR. Both analyses were conducted for ventures 1, 2, and 3, with and without additional investment for a ferromanganese plant. The target DCFROR used for the metal value determination option was 15 pct, an amount considered minimally attractive to any potential mine operator. Considering the high technical and political risks, a DCFROR of twice this amount (30 pct) would be more in line for a deep ocean mining venture.

[7]B. B. Gosling, physical scientist, Western Field Operations Center ran the computer program and provided expert advice.

Several assumptions were made in the analyses which have a material impact on results. First, financing is assumed to be 100 pct equity capital. Therefore, there are no finance charges such as capitalized interest during construction or interest expense during the subsequent operational period, when loans are normally amortized.

Because of the high degree of uncertainty in predicting reasonable values 10 to 15 yr hence, no escalation factors for costs or metal prices have been applied. The assumption is that average costs and revenues will escalate at the same rate as inflation. Included in the analyses are allowances for State and Federal income taxes, property taxes, and a 0.75-pct excise tax on gross value required by the Deep Seabed Hard Mineral Resources Act. Depletion allowances of 15 pct for copper and 22 pct for nickel, cobalt, and ferromanganese are also included.

A relatively aggressive project development schedule for all three ventures is depicted in figure 15 (see reference 30 for detailed discussion of investment timing). Research and development, prospecting, and exploration costs before year 1 are previously written off and not included in the financial analyses. Because of the requirement for extraordinary physical detail, minesite exploration would begin immediately and would continue through year 29, except for years 7 through 9 when the exploration ship and crew would participate in full-scale mining tests. Plant construction would occur in years 5 through 8 and ship construction would take place in years 6 through 9. Modifications developed during full-scale testing of the first ship would be incorporated in the second ship. Partial production would begin in year 10, and full production would be scheduled for years 12 through 30. For financial purposes, exploration cost is considered a capital cost prior to startup and an operating cost thereafter.

Table 17 shows total revenues derived from each venture based on the following prices, in January 1981 dollars:

Nickel:	
Per kilogram............	$7.72
Per pound...............	3.50
Copper:	
Per kilogram............	1.97
Per pound...............	.89
Cobalt:	
Per kilogram............	15.44
Per pound...............	7.00
Ferromanganese (78 pct Mn):	
Per metric ton..........	502.00
Per long ton............	510.00

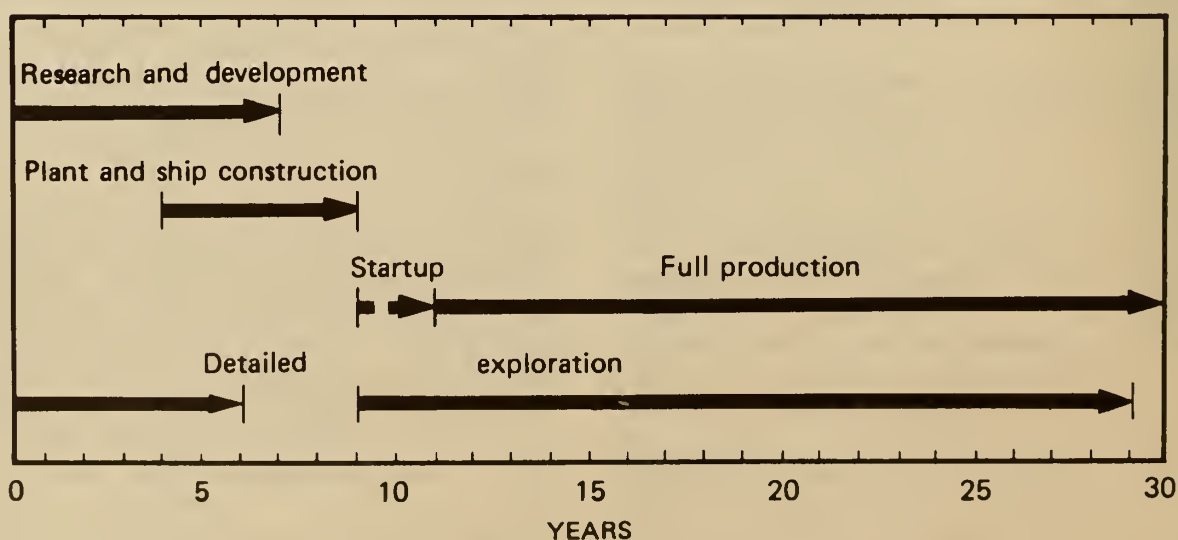

FIGURE 15. - Project development schedule.

TABLE 17. - Total commodity revenues, ventures 1, 2, and 3, million January 1981 dollars

	1	2	3
Nickel	$5,309	$4,737	$5,432
Copper	1,042	1,034	1,084
Cobalt	1,212	1,154	1,500
3-metal total	7,563	6,925	8,016
Ferromanganese (78 pct Mn)	2,052	1,606	1,978
4-metal total	9,615	8,531	9,994

The January 1981 cobalt price was considered artificially high; therefore, it was set at twice the nickel price--the approximate historical (1957-76) ratio between the prices.

Nickel, by far the most important commodity, would produce 54 to 56 pct of the revenue for four-metal operations and 68 to 70 pct for three-metal plants. If, despite the significant increase in supply, cobalt prices were to remain high, cobalt would rival nickel as the major revenue producer.

Discounted cash flow rate of return forecasts by the Bureau's MINSIM4 program are shown in table 18. All results are based on descriptions, costs, and scheduling contained in this and previous sections. Projected rates of return are very low. Venture 3, three-metal operation, is potentially the the most profitable, followed by venture 3, four-metal operation. Interestingly enough, inclusion of a ferromanganese plant decreases the apparent profitability of all three ventures.

TABLE 18. - Projected rates of return, ventures 1, 2, and 3, percent

Venture	3-metal process	4-metal process
1	4.1	3.5
2	5.0	2.7
3	6.0	5.2

Additional MINSIM4 runs were completed for each venture to determine revenue increases required to produce a DCFROR of 15 pct. In general, a 60- to 80-pct increase in total revenue would be required to achieve this modest return on capital for three- and four-metal operations. It is doubtful that such a real gain in metal prices will occur in the near future.

PRODUCTION AND SUPPLY

In all likelihood, no significant production of manganese nodules will occur in the foreseeable future. Economic, political, and, to a lesser extent, technical factors combine to exert a large negative influence. Poor economics, a situation illustrated by the financial analyses in the previous section, is largely the result of depressed metal markets and high energy, labor, and material costs.

Political uncertainty continues because the longstanding questions of rights and ownership of deep sea mineral resources continue unresolved despite passage of the Law of the Sea (LOS) Treaty. A total of 47 countries, including the United States, United Kingdom, Federal Republic of Germany, and Japan, did not sign the treaty even though certain "grandfather" rights were accorded existing ocean mining consortia. The principal objection by industrialized nations is their lack of meaningful participation in future planning and regulatory processes. Specifically, resource development, production quotas, technology transfer, licensing, and taxation would be virtually controlled by developing countries. U.S. legislation (Public Law 96-283), establishing a domestic regulatory regime for mining deep ocean minerals, is presently being used as a basis for negotiations

between nonsignatories. The intent is to establish a series of reciprocating states agreements through which mining could take place outside the LOS Treaty.

The major technological problem is whether full mine production can be reached and sustained for the life of the project. Collector effectiveness and reliability are probably the biggest unknowns. Certain aspects of ore handling and processing will require additional research and development, but appear to be well within current technological capabilities.

If, in spite of the aforementioned problems, development does occur, then the U.S. supply position of certain strategic metals would improve materially. Information in table 19 illustrates this point. U.S. consumption of nickel, copper, cobalt, and manganese in 1978 is listed, as well as projected consumption in years 1990 and 2000. The significant 1978 reliance on imports of nickel, cobalt, and manganese is also indicated. This reliance is expected to continue at a high level. Inspection of annual production figures from proposed ventures 1, 2, and 3 show that just one operation would produce from 12 to 13 pct of the projected nickel requirements, 31 to 41 pct of cobalt needs, and 10 pct of the manganese requirements in 1990, and somewhat less of projected needs in year 2000. Production from all three

TABLE 19. - Comparison of U.S. consumption[1] of nickel, copper, cobalt, and manganese with potential production from ventures 1, 2, and 3

	Ni	Cu	Co	Mn
Consumption, thousand t:				
Actual, 1978	163.7	1,879	8.8	1,236
Projected, 1990	272.2	2,500	12.5	1,615
Projected, 2000	399.2	3,200	15.9	1,814
Import reliance, 1978 ... pct	80	20	95	97
Potential annual production,[2] thousand t:				
Venture 1	35.9	27.6	4.1	163.8
Venture 2	32.0	27.4	3.9	163.8
Venture 3	36.7	28.7	5.1	163.8
Total	104.6	83.7	13.1	491.4
Amount of projected consumption supplied, pct:				
1990:				
Venture 1	13	1	33	10
Venture 2	12	1	31	10
Venture 3	13	1	41	10
Total	38	3	105	30
2000:				
Venture 1	9	<1	26	9
Venture 2	8	<1	24	9
Venture 3	9	<1	32	9
Total	26	<3	82	27

[1]Consumption figures do not include utilization of recycled metals, only primary production. Projected (46) annual growth of primary usage from a 1978 base is as follows, in percent: Ni, 3.7; Cu, 2.4; Co, 2.5; Mn, 1.4.

[2]Venture production based on the following recoveries, in percent: Ni, 92; Cu, 92; Co, 65. Production of Mn for each venture would be 163,800 t (210,000 t ferromanganese (78 pct Mn)).

Source: References 9, 32, 40, 42, and 46.

ventures would probably eliminate need for imported cobalt, and drastically reduce U.S. reliance on foreign nickel and manganese, while having a very minor effect on the domestic copper industry.

SUMMARY

Based on analysis of available resource data for study areas in the northeast Pacific Ocean, three subareas appear to contain manganese nodule deposits with the best potential for economic mining. These encompass or are adjacent to the three DOMES Sites and are designated subareas AII, BIII, and CI. Subarea AII (36,000 km^2) contains an estimated 67.0 million dry t of recoverable resource, with a grade of 1.30 wt pct nickel, 1.00 wt pct copper, and 0.21 wt pct cobalt. Subarea BIII (64,600 km^2) contains an estimated 66.9 million dry t of recoverable nodules, with an apparent grade of 1.45 wt pct nickel, 1.24 wt pct copper, and 0.25 wt pct cobalt. Subarea CI (57,600 km^2) contains an estimated recoverable resource of 148.8 million dry t of nodules grading 1.33 wt pct nickel, 1.04 wt pct copper, and 0.26 wt pct cobalt. Estimated manganese grade for the three subareas ranges between 26.8 and 27.8 wt pct.

Costing of the proposed system to mine, transport, and beneficiate the nodule ore, indicates initial investments and operation costs will be large. For three-metal recovery of nickel, copper, and cobalt, anticipated capital investments range from \$1.5 billion to \$1.7 billion (January 1981 costs); estimated operating costs are from \$71 to \$83 per metric ton of ore. If manganese is also recovered in an optional 1.4 million t/yr ferromanganese plant, estimated capital costs would increase nearly \$130 million; operating costs would add an additional \$32 to \$40 per metric ton of nodules mined.

It may be difficult to significantly reduce costs of the system as conceived and described. Scaling down capacity, purchasing power for the Cuprion plant rather than installing generators, or switching from coal- to oil-burning equipment would reduce capital requirements, but would raise unit operating costs. Transportation costs, both capital and operating, would be lower if processing were carried out closer to the minesite possibly on the island of Hawaii. Fewer transport vessels would be needed and distances traveled would be shorter. However, increased land and energy costs could partially offset savings. Operating costs, particularly those associated with mining, could be lowered as experience is gained.

Financial analyses of proposed operations predict discounted cash flow rates of return ranging from 2.7 to 6.0 pct. Operations with ferromanganese recovery may be slightly less profitable than those without. At best, these rates of return are only a fifth of the approximately 30-pct return that might be needed to attract venture capital. Presently, it is difficult to envision any manganese nodule operation, based on current technology and economics, that can realize more than a marginally acceptable profit.

If mining of nodules does occur in the near future, a significant lessening of U.S. reliance on imported nickel, cobalt, and manganese would be achieved. Just one proposed venture could supply the following percentages of projected U.S. demands for 1990: nickel, 12 to 13; cobalt, 31 to 41; and manganese, 10. Production from all three ventures would drastically reduce dependence on imported nickel and manganese and essentially eliminate the need for foreign cobalt. Detrimental effects on the domestic copper industry would be negligible.

REFERENCES

1. Agarwal, J. C., H. E. Barner, N. Beecher, D. S. Davies, and R. N. Kust. Kennecott Process for Recovery of Copper, Nickel, Cobalt, and Molybdenum From Ocean Nodules. Min. Eng., v. 31, 1979, pp. 1704-1707.

2. Agarwal, J. C., N. Beecher, D. S. Davies, G. L. Hubred, V. K. Kakaria, and R. N. Kust. Processing of Ocean Nodules: A Technical and Economic Review. J. Met., v. 28, No. 4, 1976, pp. 24-31.

3. American Association of Petroleum Geologists. Geographic Map of the Circum-Pacific Region-Northeast Quadrant. Tulsa, OK, 1977.

4. Andrews, B. V. Relative Costs of U.S. and Foreign Nodule Transport Ships (contract 7-13775, B. V. Andrews Transportation Consultant). U.S. Dept. of Commerce-NOAA, Office of Marine Minerals, Rockville, MD, 1978, 70 pp.; NTIS PB 283194.

5. Andrews, B. V. (B. V. Andrews Transportation Consultant). Private communication, 1981; available upon request from C. T. Hillman, BuMines, Spokane, WA.

6. Arthur D. Little, Inc. Technological and Economic Assessment of Manganese Nodule Mining and Processing (Revised) (contract 14-01-0001-2114). U.S. Dept. of the Interior, Office of Minerals Policy and Research Analysis, Washington, DC, 1979, 75 pp.

7. Brown, F. (E.I.C. Corp.). Private communication, 1981; available upon request from C. T. Hillman, BuMines, Spokane, WA.

8. Dames & Moore, and E.I.C. Corporation. Description of Manganese Nodule Processing Activities for Environment Studies, Vol. III. Processing Systems Technical Analysis (contract 6-35331). U.S. Dept. of Commerce-NOAA, Office of Marine Minerals, Rockville, MD, 1977, 540 pp.; NTIS PB 274912 (set).

9. DeHuff, G. L., and T. S. Jones. Manganese. Ch. in Mineral Facts and Problems, 1980 Edition. BuMines B 671, 1981, pp. 549-562.

10. Felix, D. Some Problems in Making Nodule Abundance Estimates From Seafloor Photographs. Marine Min., v. 2, 1980, pp. 293-302.

11. Fewkes, R. H., W. D. McFarland, W. R. Reinhart, and R. K. Sorem. Development of a Reliable Method for Evaluation of Deep Sea Manganese Nodule Deposits (grant G0274013, WA State Univ.). BuMines OFR 64-80, 1979, 94 pp.; NTIS PB 80-182116.

12. ______. Evaluation of Metal Resources at and Near Proposed Deep Sea Mine Sites (grant G0284008, WA State Univ.). BuMines OFR 108-80, 1980, 239 pp.; NTIS PB 80-228992.

13. Fewkes, R. H., W. D. McFarland, and R. K. Sorem. Manganese Nodule Resource Data, Sea Scope Expedition (grant G0284008, WA State Univ.). BuMines OFR 144-81, 1981; NTIS PB 82-142571.

14. Fisk, M. B., J. Z. Frazer, J. S. Elliott, and L. L. Wilson. Availability of Copper, Nickel, Cobalt, and Manganese From Ocean Ferromanganese Nodules (II) (grant G0254024, Scripps Inst. Oceanography). BuMines OFR 140(1)-80, 1979, 63 pp.; NTIS PB 81-145963.

15. Flipse, J. E. An Engineering Approach to Ocean Mining. Pres. at Offshore Tech. Conf., Houston, TX, May 1969, Paper 1035, 15 pp.

16. ______. Deep Ocean Mining Pollution Mitigation. Pres. at Offshore Tech. Conf., Houston, TX, May 1980, Paper 3834, 5 pp.

17. Flipse, J. E. (J. E. Flipse Marine Mining Consultant). Private communication, 1981; available upon request from C. T. Hillman, BuMines, Spokane, WA.

18. Frazer, J. Z. Manganese Nodule Reserves: An Updated Estimate. Marine Min., v. 1, 1977, pp. 103-123.

19. Frazer, J. Z., and G. Arrhenius. World-wide Distribution of Ferromanganese Nodules and Element Concentrations in Selected Pacific Ocean Nodules. U.S. National Science Foundation-IDOE, Washington, DC, 1972, 51 pp.; NTIS PB 234011.

20. Frazer, J. Z., and M. B. Fisk. Availability of Copper, Nickel, Cobalt, and Manganese From Ocean Ferromanganese Nodules (III) (grant G0264024, Scripps Inst. Oceanography). BuMines OFR 140 (2)-80, 1979, 112 pp.; NTIS PB 81-145971.

21. _____. Geological Factors Related to Characteristics of Seafloor Manganese Nodule Deposits (grant G0264024, Scripps Inst. Oceanography). BuMines OFR 142-80, 1980, 41 pp.; NTIS PB 81-145831.

22. Frazer, J. Z., M. B. Fisk, J. Elliott, M. White, and L. Wilson. Availability of Copper, Nickel, Cobalt, and Manganese From Ocean Ferromanganese Nodules (grant G0264024, Scripps Inst. Oceanography). BuMines OFR 121-79, 1978, 141 pp.; NTIS PB 300 356.

23. Gale, G. D. (U.S. Bureau of Mines). Private communication, 1981; available upon request from C. T. Hillman, BuMines, Spokane, WA.

24. Grote, P. B., and W. Gayman. A Technical Basis for the Development of Deep Ocean Mining Regulations (contract J0177131, Sci. Applications, Inc.). BuMines OFR 87-80, 1979, 310 pp.

25. Heezen, B. C., and M. Tharp. Bathymetric and Nodule Assessment Map Series, Northeast Equatorial Pacific Ocean. U.S. Geol. Surv. Misc. Invest. Series. Maps I-1094 A - I-1094 O, 1978.

26. Hillman, C. T. Manganese Nodule Resources of Site C, Northeast Equatorial Pacific Ocean. Unpublished BuMines report, 1980, 43 pp.; available for consultation at Western Field Operations Center, Spokane, WA.

27. Hillman, C. T., and R. D. Weldin. A Preliminary Evaluation of a Deep Ocean Polymetallic Nodule Deposit. Unpublished BuMines report, 1976, 38 pp.; available for consultation at Western Field Operations Center, Spokane, WA.

28. Hillman, C. T., and N. Wetzel. Manganese Nodule Resources Within the Northeast Pacific High Grade Zone. Unpublished BuMines report, 1980, 85 pp.; available for consultation at Western Field Operations Center, Spokane, WA.

29. Horn, D. R., B. M. Horn, and M. N. Delach. Sedimentary Provinces, North Pacific. Chart compiled by Lamont-Doherty Geol. Observatory, Columbia Univ., Palisades, NY, 1973.

30. Jugel, M. K. Deep Seabed Mining Industrial Development--Approaches and Their Timing. NOAA Draft Report, July 22, 1982, 15 pp.; available for consultation at Office of Marine Minerals, Rockville, MD.

31. Kollwentz, W. Prospecting and Exploration of Manganese Nodule Occurrences. Ch. in Review of the Activities; Edition 18, Manganese Nodules - Metals from the Sea. Metallgesellschaft AG, 1975, pp. 12-26.

32. Matthews, N. A., and S. F. Sibley. Nickel. Ch. in Mineral Facts and Problems, 1980 Edition. BuMines B 671, 1981, pp. 611-627.

33. Menard, H. W. Marine Geology of the Pacific. McGraw-Hill, 1964, 271 pp.

34. Moncrieff, A. G., and K. B. Smale-Adams. The Economics of First Generation Manganese Nodule Operations. Min. Congr. J., v. 60, No. 12, 1974, pp. 46-50.

35. Monhemius, A. J. The Extractive Metallurgy of Deep Sea Manganese Nodules. Ch. in Topics in Nonferrous Extractive Metallurgy. Soc. Chem. Ind., 1980, pp. 42-69.

36. Mudie, J. D., J. A. Grow, and J. S. Bessey. A Near-Bottom Survey of Lineated Abyssal Hills in the Equatorial Pacific. Ch. in Marine Geophysical Researches 1. D. Reidel Publ. Co., 1972, pp. 397-411.

37. Ozturgut, E., G . C. Anderson, R. E. Burns, J. W. Lavaelle, and S. A. Swift. Deep Ocean Mining of Manganese in the North Pacific, Premining Environmental Conditions and Anticipated Mining Effects. U.S. Dept. of Commerce-NOAA Tech. Memorandum ERL MESA-33, 1978, 133 pp.; available upon request from C. T. Hillman, BuMines, Spokane, WA.

38. Piper, D. Z. (comp.). Deep Ocean Environmental Study: Geology and Geochemistry of DOMES Sites A, B, and C, Equatorial North Pacific. U.S. Geol. Surv. OFR 77-778, 1977, pp. 217-266; available for consultation at U.S. Geological Survey libraries in Menlo Park, CA, Golden, CO, and Reston, VA.

39. Ryan, W. B. T., and B. C. Heezen. Smothering of Deep Sea Benthic Communities From Natural Disasters. 1976, 132 pp.; NTIS PB 279527/AS.

40. Schroeder, H. J., and J. H. Jolly. Copper. Ch. in Mineral Facts and Problems, 1980 Edition. BuMines B 671, 1981, pp. 227-244.

41. Shaw, J. L. The Economics of an Ocean Mining Project. Pres. at 4th Internat. Ocean Development Conf., Tokyo, Japan, 1976, 14 pp.; available upon request from C. T. Hillman, BuMines, Spokane, WA.

42. Sibley, S. F. Cobalt. Ch. in Mineral Facts and Problems, 1980 Edition. BuMines B 671, 1981, pp. 199-214.

43. Sorem, R. K., and R. H. Fewkes. Manganese Nodule Research Data, and Methods of Investigation. IFI/Plenum, 1979, 723 pp.

44. Sullivan, A. F., and F. A. Ruggeri. Submerged Pumps for Deep Ocean Mining. CIM Bull., v. 67, No. 752, 1974, pp. 80-85.

45. Tinsley, C. R. Manganese Nodule Mining Industry: A Study of Expected Investment Requirements. Ch. in Manganese Nodules, Dimensions, and Perspectives. D. Reidel Publ. Co., 1979, pp. 119-138.

46. U.S. Bureau of Mines. Mineral Commodity Summaries. 1981, pp. 36, 49, 94, 104.

47. _____. MINSIM4/OPEN Documentation. Unpublished internal document, 1979, 58 pp.; available upon request from B. B. Gosling, BuMines, Spokane, WA.

48. U.S. Navy. Marine Climatic Atlas of the World, Volume II, North Pacific Ocean. Director, Naval Oceanography and Meteorology, 1977, 388 pp.

49. Van Andel, T. H., G. R. Heath, and T. C. Moore. Cenozoic History and Paleoceanography of the Central Equatorial Pacific Ocean: A Regional Synthesis of Deep Sea Drilling Data. Geol. Soc. Am. Memoir 143, 1975, 134 pp.

50. Welling, C. G. Next Step in Ocean Mining--Large Scale Test. Min. Congr. J., v. 62, No. 12, 1976, pp. 46-51.

APPENDIX A.--DISCUSSION OF ABUNDANCE AND RESOURCE ESTIMATES

Abundance is defined as the weight of nodules per unit area of seafloor, and is usually given in either wet or dry kilograms per square meter. This expression may be easily converted to metric tons per square kilometer by multiplying by a factor of 1,000. Therefore, an abundance of 6 kg/m^2 would be equivalent to 6,000 t/km^2. Dry abundance is only about 70 pct of wet abundance, because nodules are extremely porous and contain approximately 30 wt pct water. Therefore, a wet nodule abundance of 10 kg/m^2 would equal 7 kg/m^2 dry.

Resource quantities are generally expressed in dry metric tons and are obtained by multiplying average dry abundance by the size (square kilometers) of the site being considered. If, for example, an area covers 10,000 km^2 and has an average wet abundance of 8.6 kg/m^2, the resource quantity would be calculated as follows:

1. Convert wet abundance to dry abundance--

$$8.6 \text{ kg/m}^2 \text{ (wet)} \times 0.70 = 6.0 \text{ kg/m}^2 \text{ (dry)}.$$

2. Convert dry abundance to metric tons per square kilometer--

$$6.0 \text{ kg/m}^2 \text{ (dry)} \times 1 \times 10^6 \text{ m}^2/\text{km}^2 = 6.0 \times 10^6 \text{ kg/km}^2 \text{ (dry)}.$$

$$6.0 \times 10^6 \text{ kg/km}^2 \div 1{,}000 \text{ kg/t} = 6{,}000 \text{ t/km}^2.$$

3. Determine resource quantity of 10,000-km^2 area--

$$6{,}000 \text{ t/km}^2 \text{ (dry)} \times 10{,}000 \text{ km}^2 = 60{,}000{,}000 \text{ t (dry)}.$$

Calculated in this manner, the total tonnage is considered a gross estimate, and should be reduced by a series of practical considerations to get a reasonable estimate of recoverable resource (see text).

Abundances, the basis for resource estimates, are derived from bottom sampling, seafloor photographs, and television video tapes. Bottom samples provide the best data. To determine abundance, recovered nodules are simply weighed and the weight is then divided by the area of seafloor sampled (a constant for each sampling device).

Box cores and certain grab samplers, such as the Okean-70 (Japanese), take a large sample and thereby afford the most accurate estimate. Also, because of specialized construction and weight of these devices, there is a low probability that portions of the sample will be lost. Additionally, box core samples are relatively undisturbed and can be used for a variety of studies. The main drawback is that use of these devices is time consuming and expensive, because they must be lowered and raised by cable. Conversely, use of free-fall devices is comparatively cheap. Many can be launched overboard (without tether) and recovered in relatively short periods of time, but there is a continuing risk that an incomplete sample will be taken or portions lost during ascent. Therefore, estimates based on grab samples are apt to be less than true abundances.

Seafloor photography and television are excellent tools for judging continuity between sample points. However, difficulties exist in converting apparent nodule populations in photographs or video tapes to abundances. One problem is that individual nodule weights must be estimated by comparing cross-sectional area with graphs developed from nearby bottom sampling. A second, more serious problem is that portions of nodules are usually obscured by a light coating of sediment. Felix (10)[1] indicates that significantly low estimates may result. Correction factors can be determined and applied if

[1]Underlined numbers in parentheses refer to items in the list of references preceding this appendix.

a detailed sampling program is carried out. No such detail is now publicly available.

Abundances in this report are based on average estimates from 185 ship stations. Because raw data were developed on a random basis, no attempt was made to differentiate between estimates based on box cores, grab samples, or photographs, nor was more significance placed on any particular estimate method. This being the case, assigned abundances and consequently resource tonnages are probably low, and are considered minimum values.

APPENDIX B.--SAMPLE STATION LOCATIONS, ABUNDANCE ESTIMATES, AND ANALYSES FOR STUDY AREAS A, B, AND C

This appendix contains a series of tables with all available sample data for each subarea. Tables B-7, B-11, and B-14 contain information from additional samples taken from locations nearby, yet outside subarea boundaries.

TABLE B-1. - Subarea AI--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
206	11.132	148.072			1.43	1.19	0.22		29.30	6.20
207	10.978	148.325			1.32	1.12	.21		25.40	6.53
208	11.030	148.382			1.48	1.12	.19		25.00	6.30
209	11.018	148.498			1.24	1.03	.20	0.05	25.00	5.44
210	11.513	148.578			1.09	.95	.17		22.40	5.30
211	11.803	148.662			1.35	1.31	.20	.06	28.70	5.78
212	11.233	148.833	NPr							
213	11.817	148.933	20-50		.75	.50	.25		20.00	9.90
214	11.167	148.950	20-50							
215	11.487	149.183			1.10	.77	.24	.02	22.00	7.62
216	12.023	149.300			1.31	1.22	.23	.03	27.30	6.45
217	11.433	149.283			.79	.50	.18		17.20	14.50
218	11.667	150.117			1.28	.95	.23		24.00	6.95
219	11.970	150.308			1.23	1.08	.19	.03	24.50	7.54
Mean					1.20	0.98	0.21	0.04	24.23	7.38
Standard deviation					0.23	0.26	0.03	0.02	3.48	2.56
Number of samples					12	12	12	5	12	12

NPr Nodules present, no additional information available.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-2. - Subarea AII--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
32	9.035	145.000			1.43	1.01	0.17	0.06	30.90	6.30
33	8.980	145.110			1.42	1.17	.18	.06	30.10	5.90
34	8.955	145.807			1.62	1.35	.17	.06	28.30	4.64
35	8.867	146.440		13.87						
36	8.902	146.443		11.25	1.47	1.08	.32		27.40	6.30
37	8.932	146.445		10.62	1.56	1.18			29.50	5.44
38	8.965	146.445		3.12						
39	9.047	146.457		4.37	1.89	1.52			32.00	3.65
40	9.050	146.487		8.75	1.52	1.06			31.00	5.66
41	8.997	146.497		18.12	1.31	.74			23.20	7.77
42	8.958	146.492		1.87	1.71	1.23			26.30	5.53
43	8.932	146.493		6.25	1.55	1.23	.17		29.40	5.20
44	8.897	146.492		14.37	1.78	1.43			30.70	5.04
45	9.057	146.542		3.75	1.91	1.60			30.40	4.02
46	9.062	146.572		9.00	1.44	1.12			26.50	4.57
47	8.965	146.668			1.35	1.22	.19	.05	29.90	5.85
48	8.740	147.532			1.47	1.22	.18	.05	30.10	5.74
49	8.727	147.590			1.48	1.27	.18	.06	29.60	5.68
50	8.575	147.778			1.30	1.25	.16	.05	27.30	5.98
51	9.100	145.300			1.52	1.27	.26	.04	26.20	5.30
52	9.447	145.555			1.44	1.14	.22	.05	28.30	5.75
53	9.657	146.332			1.33	1.08	.21		26.20	7.07
54	9.250	146.350	>50		1.24	.80	.26		22.00	6.50
55	8.590	146.300	20-50		1.50	1.20	.20		27.00	5.00
56	8.480	147.250	<20		1.30	1.15	.25		28.00	5.00
57	9.503	145.805			1.60	1.30			30.00	6.00
58	9.457	145.825			1.50	1.30			30.60	6.00
59	9.457	145.828			1.50	1.30			31.20	5.30
60	9.457	145.833			1.50	1.20			32.30	5.00
61	9.457	145.838			1.50	1.30			31.20	5.40
62	9.457	145.842			1.30	1.20			30.00	5.00
63	9.453	145.842			1.40	1.20			30.00	5.00
64	9.457	145.845			1.30	1.30			29.00	5.60
65	9.498	145.828			1.40	1.10			29.80	6.00
66	9.508	145.828			1.40	1.10			29.40	6.00
67	9.502	145.828			1.40	1.10			30.00	6.00
68	9.523	145.828			1.60	1.30			30.50	5.00
69	9.512	145.828			1.50	1.30			28.30	5.30
70	9.517	145.828			1.50	1.10			30.30	6.00
71	9.505	145.850			1.50	1.10			31.00	6.00
72	9.518	145.850			1.70	1.40			33.30	4.50
73	9.515	145.850			1.50	1.30			30.70	5.50
74	9.522	145.850			1.60	1.30			29.00	5.00
75	9.520	145.855			1.50	1.20			29.00	5.50
76	9.458	145.867			1.50	1.30			31.20	5.60
77	9.458	145.870			1.50	1.30			30.40	5.80
79	9.458	145.880			1.50	1.20			31.60	5.80
80	9.458	145.885			1.60	1.30			31.40	5.70
81	9.375	145.882			1.40	1.20			31.00	6.00
82	9.373	145.885			1.30	.90			30.00	6.20

See footnote at end of table.

TABLE B-2. - Subarea AII--location, abundance, and analytical data--Continued

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
83	9.372	145.890			1.40	1.20			31.00	6.00
84	9.372	145.893			1.50	1.20			30.60	5.50
85	9.370	145.902			1.40	1.10			30.00	6.00
86	9.370	145.905			1.40	1.00			26.90	5.40
87	9.340	145.905			1.40	1.10			31.00	6.20
88	9.340	145.910			1.40	1.10			30.00	6.80
89	9.340	145.913			1.20	.80			28.40	9.00
90	9.340	145.918			.80	.60			23.00	14.00
91	9.278	145.917			1.40	1.10			31.60	6.40
92	9.362	145.922			1.20	.90			30.00	7.50
93	9.362	145.925			1.00	.80			26.00	11.00
94	9.362	145.928			1.40	1.00			31.40	6.00
95	9.362	145.933			1.30	1.00			26.00	6.00
96	9.362	145.940			1.40	1.00			27.00	6.00
97	9.335	145.932			1.10	.70			23.00	10.00
98	9.333	145.940			1.00	.60			25.00	10.00
99	9.347	145.950			.90	.50			24.00	12.00
100	9.358	145.950			1.20	.80			27.00	9.00
101	9.352	145.950			.90	.70			26.00	10.30
102	9.365	145.950			1.20	.90			22.00	8.00
103	9.362	145.950			1.20	.80			21.00	7.00
104	9.318	145.955			1.00	.80			24.00	10.00
105	9.280	145.958			1.50	1.10			24.00	10.00
106	9.297	145.958			1.20	.90			23.00	8.00
107	9.289	145.958			1.50	1.10			28.60	7.00
108	9.315	145.972			.90	.60			25.00	13.00
109	9.315	145.977			1.10	.70			28.00	10.30
110	9.315	145.982			1.10	.70			27.00	11.00
111	9.315	145.985			1.10	.70			26.00	10.50
112	9.315	145.985			1.20	.70			26.00	10.00
113	9.315	145.998			1.00	.60			24.70	11.40
114	9.315	146.000			.90	.50			25.80	12.00
115	9.315	146.005			.80	.50			26.40	12.00
116	9.315	146.008			1.40	1.20			30.30	5.70
117	9.328	145.978			1.00	.70			26.00	11.00
118	9.337	145.983			1.30	.90			28.00	8.00
119	9.308	145.987			1.30	1.10			25.00	6.00
120	9.357	146.015			1.50	1.30			28.00	5.50
121	9.265	146.017			1.50	1.20			30.00	5.50
122	9.267	146.022			1.40	1.10			30.80	5.40
123	9.272	146.027			1.40	1.10			31.00	5.60
124	9.275	146.032			1.50	1.30			30.80	5.00
125	9.278	146.035			1.50	1.10			30.50	
126	9.280	146.040			1.40	1.00			29.50	6.00
127	9.322	146.018			1.45	1.20			31.50	5.50
128	9.315	146.020			1.50	1.30			27.50	5.00
129	9.313	146.023			1.00	.70			24.80	9.70
130	9.313	146.028			1.40	1.00			29.40	8.00
131	9.313	146.032			1.30	1.00			27.00	6.20
132	9.313	146.043			.90	.50			23.50	13.00

See footnote at end of table.

TABLE B-2. - Subarea AII--location, abundance, and analytical data--Continued

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis Ni	Cu	Co	Mo	Mn	Fe
133	9.520	146.035			1.10	1.00			25.30	4.30
134	9.268	146.042			1.40	1.10			27.00	5.80
135	9.270	146.043			1.40	1.10			29.70	5.40
136	9.273	146.047			1.40	1.20			28.00	5.50
137	9.275	146.050			1.70	1.00			29.00	7.00
138	9.278	146.055			1.30	1.00			27.80	
139	9.282	146.060			1.30	.90			27.80	
140	9.452	146.043			1.30	1.10			30.50	6.00
141	9.523	146.048			1.00	1.00			28.00	2.80
142	9.523	146.053			1.10	1.00			28.00	4.20
143	9.523	146.060			1.10	1.00			26.40	3.50
144	9.523	146.062			1.10	.90			27.30	5.00
145	9.505	146.048			1.50	1.10			29.00	5.70
146	9.515	146.050			1.30	1.10			25.00	4.00
147	9.515	146.060			1.30	1.20			27.80	4.80
148	9.515	146.063			1.30	1.20			26.40	4.70
149	9.515	146.067			1.00	.70			25.00	7.60
150	9.515	146.070			1.00	.70			25.60	8.80
151	9.503	146.062			1.30	1.00			26.40	4.80
152	9.503	146.065			1.00	.90			26.30	7.00
153	9.503	146.070			1.00	.90			26.70	7.00
154	9.503	146.080			1.00	.80			26.00	9.70
155	9.297	146.063			1.00	.80			23.80	11.40
156	9.358	146.067			1.40	1.10			31.00	6.00
157	9.305	146.073			.90	.60			23.20	10.30
158	9.308	146.077			.90	.50			24.50	13.00
159	9.312	146.082			.90	.50			25.30	12.00
160	9.315	146.085			.90	.60			25.20	12.00
161	9.290	146.073			1.00	.90			26.00	7.00
162	9.342	146.075			1.30	.90			28.00	8.40
163	9.322	146.075			.80	.50			27.00	12.00
164	9.333	146.075			.80	.50			27.30	11.70
165	9.332	146.075			.90	.60			26.30	12.70
166	9.327	146.075			.90	.50			26.20	13.00
167	9.270	146.075			1.00	1.00			26.00	7.00
168	9.273	146.075			1.40	1.00			27.20	6.50
169	9.265	146.075			1.20	.90			23.00	7.40
170	9.262	146.075			1.60	1.20			27.80	5.00
171	9.250	146.075			1.00	.70			24.40	9.30
172	9.310	146.088			.85	.50			24.50	13.30
173	9.480	146.115			1.30	1.00			28.00	6.00
		Mean..................		8.78	1.30	1.00	0.21	0.05	27.85	7.13
		Standard deviation....		NC	0.25	0.25	0.05	0.01	2.63	2.57
		Number of samples.....		12	139	139	15	9	139	136

NC Not calculated.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-3. - Subarea AIII--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
174	8.542	148.932			1.38	1.28	0.17	0.05	30.70	6.30
175	8.483	149.417	<20		1.45	1.35	.20		26.80	5.90
176	8.250	149.500	20-50		1.15	.85	.26		22.00	7.50
177	8.450	149.783	47.2	8.30						
178	8.417	149.800	>50							
179	8.200	150.300			1.25	1.15	.25		25.00	5.50
180	8.400	150.283			1.35	1.25	.20		26.80	6.90
181	8.450	150.283			1.45	1.30	.20		27.60	6.30
182	8.483	150.317	>50							
183	8.483	150.400	.7	.20						
184	8.453	150.478	<20							
185	7.750	150.683			1.44	1.54	.20		26.67	6.17
186	8.083	150.683	<20							
187	7.817	150.758	<20							
188	8.033	150.733			1.35	1.40	.20		29.80	6.40
189	8.150	150.717			1.30	1.15			26.50	7.50
190	8.197	150.738	40.9	7.50						
191	8.475	150.742	>50	13.70	1.08	.77	.26	.04	23.93	10.08
192	8.217	150.767	NOb							
193	8.133	150.767	<20		.78	.59	.42		20.81	13.73
194	8.250	150.800	20-50							
195	8.458	150.778	<20	.60	1.08	.78	.25		24.62	10.29
196	8.503	150.797	20-50	11.50	1.45	1.23	.21		27.01	5.99
197	8.467	150.800			1.44	1.45	.18		27.33	4.92
198	8.400	150.800	2.4	.60						
199	8.500	150.833	>50		.85	.70	.23		20.40	11.00
200	8.457	150.837	>50	12.30	1.38	1.04	.24		27.23	7.66
201	8.417	150.817	<20		1.39	1.35	.21		27.90	6.50
202	8.417	150.900	<20		1.42	1.63	.17		29.14	5.02
203	8.433	150.917	4.9	1.20						
204	8.273	150.933			1.47	1.34	.21	.05	28.75	6.72
205	7.825	150.950	>50							
		Mean..................		6.21	1.28	1.16	0.22	0.05	26.26	7.38
		Standard deviation....		NC	0.21	0.30	0.06	0.01	2.70	2.30
		Number of samples.....		9	19	19	18	3	19	19

NC Not calculated.
NOb No nodules observed in photographs or no nodules recovered in samples.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-4. - Subarea AIV--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
1	9.900	149.950			0.75	0.39	0.20		22.65	14.23
2	9.717	149.950			1.08	.65	.21		23.00	7.70
3	9.683	149.900			1.27	.82	.26		22.00	7.79
4	9.220	149.817			.90	.66	.28		17.00	10.10
5	8.783	149.867			.75	.50	.30		22.00	11.80
6	8.833	149.917			1.02	.65	.28		17.80	7.92
7	8.718	150.235		10.50	.99	.69	.33		23.35	11.64
8	8.685	150.252		5.50	.89	.54	.36		22.90	11.78
9	8.690	150.250		21.70	.99	.58	.24		20.90	11.48
10	8.695	150.312		11.40	.90	.62	.31		22.30	11.60
11	8.730	150.312		19.90	.87	.64	.27		21.50	11.11
12	9.200	150.375			1.00	.80	.25		17.00	5.00
13	9.333	150.583			.55	.43	.17		17.20	14.00
14	9.450	150.700			.33	.33	.80		4.50	9.40
15	8.817	150.783	45.6	8.20						
16	9.333	150.807		6.20	.78	.59	.42		20.80	13.73
17	9.345	150.845		4.70	1.54	1.10	.22		23.80	8.14
18	9.400	150.832		6.10	1.30	.86	.24		21.40	10.21
19	9.318	150.840		9.70	1.33	1.00	.22		24.90	8.82
20	9.450	150.817	46.9	8.10						
21	9.423	150.845	>50							
22	9.327	150.845		.60	1.20	.83	.26		19.60	6.78
23	9.355	150.857		.10						
24	9.317	150.863	58.0	8.80						
25	9.317	150.875		10.20	.86	.59	.32		22.30	11.48
26	9.397	150.880		15.20	.82	.55	.26		20.90	11.13
27	9.500	150.883			.77	.53	.25		15.67	8.80
28	9.533	150.900	<20							
29	9.517	150.950			.83	.59	.19		15.80	8.15
30	9.917	150.933			.95	.70	.25		23.60	10.30
31	9.883	150.950			1.25	1.00	.20		24.40	7.10
241	10.000	150.000			.92	.66	.25		19.00	9.60
		Mean..................		9.18	0.96	0.67	0.28		20.24	9.99
		Standard deviation....		NC	0.26	0.19	0.12		4.19	2.33
		Number of samples.....		16	26	26	26	0	26	26

NC Not calculated.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-5. - Subarea AV--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
250	8.308	151.023	NOb							
252	8.050	151.400	<20		1.30	1.20	0.25		24.00	6.00
253	8.275	151.122	20-50	8.10	1.31	1.09	.24		27.00	7.70
254	8.302	151.158	<20	.20						
255	8.267	151.180	33.0	5.90						
256	8.267	151.188	<20	1.70	1.61	1.64	.18		30.00	4.73
257	8.293	151.220	20-50	.70	1.36	1.33	.20		25.23	5.82
258	8.238	151.238	20-50	9.40	1.34	1.06	.22		27.30	8.91
259	8.475	151.152	NPr							
260	8.467	151.150			1.19	1.05	.18		25.40	7.30
261	9.005	151.182		.10						
262	9.027	151.203	6.5	1.60						
263	9.038	151.187	<20	1.60	1.53	1.45	.17		25.13	4.92
264	9.038	151.190	<20							
265	9.042	151.177	<20	1.60	1.62	1.56	.26		30.40	3.78
266	9.053	151.183		3.80						
267	9.058	151.185	<20	4.90	1.46	1.31	.22		24.22	5.57
268	9.080	151.157	<20	.70						
269	9.080	151.185	<20	7.60	1.72	1.55	.27		29.90	4.25
270	9.065	151.237	<20	3.60	1.59	1.49	.29		29.50	4.66
271	9.100	151.500	<20		1.30	1.10	.25		19.00	5.50
272	9.033	151.567			1.25	1.06	.15		26.80	5.30
273	8.400	151.800	NOb							
274	8.417	151.883			1.36	1.18	.18		23.30	4.81
275	9.367	151.967	<20		1.28	.09	.17		25.30	4.25
276	9.450	152.017	>50							
277	9.383	152.033			1.30	1.05	.20		26.80	7.50
		Mean..................		3.43	1.41	1.20	0.21		26.20	5.69
		Standard deviation....		NC	0.16	0.36	0.04		2.96	1.45
		Number of samples.....		15	16	16	16	0	16	16

NC Not calculated.
NOb No nodules observed in photographs or no nodules recovered in samples.
NPr Nodules present, no additional information available.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-6. - Subarea AVI--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
278	8.033	152.150	20-50							
279	7.950	152.250	20-50		1.20	1.10	0.25		22.50	7.00
280	7.970	152.287			1.24	1.08	.24		23.01	7.27
281	8.050	152.300	NOb							
282	8.133	152.267	>50		.84	.55	.21		19.45	9.55
283	8.098	152.603			.96	.72	.22		19.86	9.24
284	7.945	152.805			1.35	1.32	.15		26.10	5.25
293	8.342	152.952	NOb							
294	8.321	152.959			1.11	1.20	.14	0.04	26.40	5.74
295	8.314	152.961	NOb							
296	8.326	152.967	NOb							
297	8.305	152.964			.90	1.03	.15	.04	26.40	6.47
298	8.296	152.965			1.08	.93	.18	.04	24.60	7.94
299	8.281	152.968			1.01	.99	.16	.03	22.40	6.56
300	8.272	152.970			1.20	1.25	.21	.04	24.41	7.79
301	8.264	152.972			1.01	.86	.20	.03	24.00	9.24
302	8.366	152.986	NOb							
303	8.363	152.986	NOb							
304	8.371	152.988	NOb							
305	8.380	152.989	NPr							
306	8.375	152.989			1.28	1.65	.13	.04	25.90	5.44
307	8.390	152.991	NOb							
308	8.394	152.992			.90	1.03	.15	.04	26.40	6.47
309	8.400	153.000	NPr							
310	8.285	153.009	NOb							
311	8.321	153.009	NOb							
312	8.347	153.016			1.02	.99	.18	.03	23.30	7.49
313	8.382	153.026	NOb							
314	8.337	153.027	NOb							
315	8.326	153.027	NOb							
316	8.326	153.027	NOb							
317	8.328	153.031	NOb							
318	8.324	153.035	NOb							
319	8.332	153.031			1.05	1.05	.17	.04	25.70	6.37
320	8.369	153.032			.94	.90	.17	.03	22.50	7.59
321	8.319	153.037			.92	.99	.16	.04	24.40	7.28
322	8.315	153.040			1.07	1.09	.16	.04	26.60	6.93
323	8.351	153.038								
324	8.381	153.039			1.12	1.21	.14	.04	25.90	5.98
325	8.306	153.047			1.59	1.44	.19	.03	25.30	5.77
326	8.380	153.050	NOb							
327	8.383	153.056	NOb							
328	8.381	153.062	NOb							
329	8.383	153.069	NOb							
333	7.885	153.550	<20		1.45	1.39	.09		26.70	4.83
334	8.417	153.450	20-50		1.49	1.40	.22		23.40	4.59
340	8.095	153.915			1.16	1.15	.15	.05	23.90	7.75
		Mean............................			1.13	1.10	0.17	0.04	24.31	6.89
		Standard deviation................			0.20	0.24	0.04	0.01	2.05	1.36
		Number of samples.................			23	23	23	16	23	23

NOb No nodules observed in photographs or no nodules recovered in samples.
NPr Nodules present, no additional information available.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available

TABLE B-7. - Study area A--location, abundance, and analytical data outside subareas

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
221	11.933	146.448			1.31	1.03	0.23	0.04	26.70	6.45
222	9.133	146.817			1.00	.75	.20		19.60	7.00
223	9.450	147.300	20-50		.70	.40	.35		20.00	12.00
224	9.083	148.750			1.04	.75	.21		21.60	6.80
225	9.350	148.733	<20	22.20	1.35	1.02	.24		26.80	7.30
226	10.283	148.700	>50		1.20	.86	.24		27.10	7.55
227	10.283	148.750			1.40	1.05	.19		28.00	6.50
228	7.400	149.033			1.45	1.30	.15		29.60	5.20
229	7.700	149.050	<20							
230	7.750	149.100	<20		1.00	.75	.25		22.00	10.20
231	8.875	149.100	<20							
232	9.700	149.250	<20							
233	9.275	149.400			1.35	1.25	.25		19.00	5.00
234	9.650	149.400			1.28	1.16	.16		26.10	6.10
235	9.683	149.517	<20							
236	10.350	149.450	<20		1.00	.70	.30		21.50	9.50
237	12.717	149.067			1.10	.95	.30		24.00	8.90
242	10.633	150.033	20-50		.95	.60	.20		18.80	7.20
243	10.300	150.500	>50		.50	.40	.40		18.00	11.00
244	12.650	150.167			.58	.35	.35		21.07	14.43
245	12.600	150.233	>50		.75	.45	.40		22.80	14.10
246	7.567	150.700	NOb							
247	7.450	150.767	<20		1.42	1.68	.15		27.90	5.58
248	7.417	150.833	NPr							
249	9.900	151.042	>50							
251	7.433	151.783			1.43	1.39	.15		25.73	4.60
285	8.663	152.630		.62						
286	8.663	152.630		.12						
287	8.663	152.630		.25						
288	8.950	152.867			.66	.61	.22		17.02	8.84
289	8.950	152.867			1.33	.41	.78		22.61	10.94
290	9.983	152.950		5.62						
291	9.983	152.950		6.25						
292	9.983	152.950			1.53	1.11			29.40	6.52
330	8.933	153.083	<20		.95	.80	.20		24.50	8.50
331	9.900	153.117	<20							
332	7.350	153.200	<20							
335	9.700	153.550	<20		1.00	.60	.35		23.00	13.80
336	9.733	153.550	<20		.67	.39	.34		24.10	15.90
337	9.733	153.600	>50		.84	.52	.34		21.40	12.64
338	9.717	153.617	>50							
339	9.617	153.697			1.37	1.59	.13		25.45	5.76
341	11.367	153.617		.25	1.33	1.12	.12		20.20	5.00
342	11.367	153.617		.25	1.33	1.14	.14		21.80	5.10
343	11.367	153.617		1.88	1.38	1.22	.14		25.30	5.85
344	10.935	153.307	>50	22.20	.52	.26	.24		19.80	12.51
345	11.010	153.333	<20	.40	1.16	1.02	.16		23.90	5.86
346	10.968	153.418	<20	4.00	1.31	1.08	.15		25.20	5.60
347	11.007	153.435	<20	1.80	1.15	1.08	.10		25.60	5.75
348	10.947	153.445	>50	10.80	.75	.44	.24		19.10	11.10
349	11.005	153.470	20-50	8.00	.90	.45	.22		18.80	8.56
350	11.013	153.478	20-50	7.00						
351	10.977	153.477	20-50	12.00						
352	10.993	153.480	20-50	6.40	.73	.49	.15		19.00	9.72
353	10.860	153.497	>50	14.00	.78	.50	.26		19.60	11.94
354	11.110	153.522		1.80						
		Mean....................		5.45	1.07	0.83	0.24	0.04	22.95	8.56
		Standard deviation......		NC	0.30	0.37	0.12	NC	3.41	3.15
		Number of samples.......		19	38	38	37	1	38	38

NC Not calculated. NOb No nodules observed in photographs or no nodules recovered in samples.
NPr Nodules present, no additional information available.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-8. - Subarea BI--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
523	13.922	139.772			1.26	1.26	0.21		30.00	6.45
524	13.928	139.780			1.31	1.39	.19		27.50	3.95
525	13.767	140.017			1.16	.94	.14		23.15	6.65
526	13.750	140.050			1.20	.85	.21		19.73	5.40
527	13.887	140.057			1.42	1.32	.22		30.40	4.20
528	13.667	140.067			1.07	.77	.24		22.20	7.37
529	13.733	140.083			1.00	.73	.24		17.00	7.80
530	13.717	140.100			.97	.68	.21		16.80	6.30
531	13.717	140.133			1.00	.64	.16		15.30	4.60
532	13.700	140.150			1.32	.90	.44		27.70	5.77
533	13.700	140.167			1.31	1.17	.17		26.80	5.60
534	13.700	140.200			1.41	1.22	.20		29.90	5.50
535	13.683	140.233			1.40	1.22	.19		28.50	5.00
536	13.683	140.283			1.36	1.00	.43		36.40	5.17
537	13.683	140.333			1.35	1.08	.41		28.90	5.10
538	13.683	140.333			1.41	1.14	.14		20.80	6.40
539	13.683	140.383			1.63	1.55	.10		31.70	4.96
540	13.667	140.433			1.45	1.46	.10		31.00	4.58
541	13.650	140.450			.88	.80	.23		20.20	5.30
542	13.650	140.467			1.07	.79	.44		26.30	5.79
543	13.647	140.470			1.09	1.06	.21		22.90	4.93
544	13.617	140.500			.86	.56	.14		18.20	7.20
545	13.633	140.500			1.43	1.20	.22		29.60	5.11
546	13.617	140.517			.32	.24	.14		7.50	5.20
547	13.617	140.533			1.37	1.10	.23		23.80	5.00
548	13.600	140.550			.88	.60	.24		18.10	10.70
549	13.617	140.550			.77	.62	.15		15.80	6.50
550	13.600	140.567			.86	.71	.20		15.50	5.80
551	13.600	140.583			1.16	.86	.20		23.60	6.30
552	13.583	140.600			1.02	.76	.16		16.30	4.80
553	13.583	140.617			.77	.45	.24		19.90	4.84
554	13.567	140.617			1.30	1.02	.23		27.60	6.50
555	13.583	140.617			1.16	.88	.18		21.40	7.80
556	13.569	140.633			.89	.48	.52		22.00	9.24
557	13.567	140.633			1.02	.77	.22		21.10	7.60
558	13.567	140.650			1.58	1.08	.18		20.80	9.20
559	13.550	140.667			1.21	.74	.43		25.70	5.58
560	13.550	140.667			1.02	.82	.23		20.20	5.40
561	13.550	140.683			1.30	1.08	.12		25.70	5.16
562	13.550	140.683			1.04	.56	.44		23.60	6.98
563	13.517	140.717			.97	.83	.16		19.20	5.90
564	13.533	140.717			1.24	.89	.16		21.30	5.45
565	13.500	140.717			1.02	.64	.18		14.60	5.30
566	13.567	140.717			1.06	.87	.24		19.10	5.60
567	13.483	140.733			.92	.58	.18		14.30	5.20
568	13.467	140.733			.94	.61	.21		19.40	6.75
569	13.400	140.750			.94	.64	.28		18.30	7.40
570	13.433	140.750			.90	.64	.18		18.40	7.50
571	13.417	140.750			.95	.72	.22		18.10	6.10
572	13.400	140.767			.97	.63	.27		18.50	9.90
573	13.483	140.750			1.04	.49	.46		22.80	6.63
574	13.450	140.750			.90	.52	.22		19.20	6.29
575	13.450	140.750			.86	.61	.23		16.40	6.00
576	13.483	140.750			1.33	.95	.24		24.20	6.00
577	13.467	140.750			1.07	.81	.21		22.10	5.35
578	13.467	140.750			.67	.52	.13		12.90	5.50
579	13.233	140.833			.61	.29	.42		27.60	5.11
		Mean................................			1.10	0.84	0.23		22.03	6.09
		Standard deviation..................			0.25	0.29	0.10		5.54	1.38
		Number of samples...................			57	57	57		57	56

[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-9. - Subarea BII--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis Ni	Cu	Co	Mo	Mn	Fe
580	13.160	141.008			0.57	0.39	0.20		22.65	12.70
581	13.003	141.213			1.46	1.20	.26		28.80	4.75
582	12.850	141.583			1.00	.73	.12		16.80	5.80
583	12.833	141.583			.92	.51	.30		20.90	8.38
584	12.850	141.617			.94	.64	.28		17.90	7.40
585	12.850	141.617			1.12	.83	.24			6.61
586	12.833	141.617			1.25	.96	.21		23.80	5.40
587	12.817	141.633			1.36	1.07	.20		27.80	4.96
588	12.817	141.667			1.33	1.05	.38		28.70	5.60
589	12.817	141.667			1.43	1.11	.22		30.50	4.84
590	12.817	141.683			1.51	1.16	.23		32.00	5.50
591	12.800	141.700			1.39	1.17	.21		30.20	4.80
592	12.800	141.717			1.63	1.29	.18		24.80	7.70
593	12.800	141.717			1.43	1.08	.19		30.70	4.19
594	12.783	141.733			1.14	.91	.26		23.40	5.40
595	12.783	141.733			1.39	1.06	.20		30.50	4.68
596	12.767	141.750			.88	.57	.18		18.80	6.35
597	12.767	141.750			1.39	1.22	.19		29.20	6.16
598	12.767	141.767			1.10	.79	.16		23.60	6.80
599	12.750	141.783			1.24	.92	.25		22.60	5.70
600	12.683	141.783			1.03	.78	.24		19.80	5.90
601	12.733	141.800			.97	.69	.24		18.90	11.90
602	12.733	141.817			1.32	1.07	.14		24.70	5.96
603	12.717	141.842			1.43	1.26	.13		29.10	6.24
604	12.717	141.850			.84	.71	.29		18.50	5.50
605	12.717	141.850			1.37	1.35	.20		21.50	4.70
606	12.700	141.883			1.23	.95	.22		21.80	5.50
607	12.667	141.900			1.18	.72	.22		24.30	5.82
608	12.667	141.917			.86	.61	.24		14.90	4.80
609	12.667	141.933			1.21	.92	.27		25.20	8.10
610	12.650	141.950			1.18	.95	.23		26.00	6.20
611	12.633	141.967			.93	.52	.27		19.40	9.30
612	12.633	142.000			1.48	1.33	.12		30.60	5.22
613	12.617	142.017			.98	.71	.22		18.20	5.40
614	12.600	142.058			1.30	1.05	.09		26.40	6.60
615	12.600	142.067			1.02	.81	.25		20.40	5.80
616	12.667	142.068			.83	.58	.25		23.80	6.30
617	12.583	142.150			1.25	1.21	.21	0.06	31.30	4.91
Mean					1.18	0.92	0.23	0.06	24.28	6.26
Standard deviation					0.24	0.26	0.07	NC	4.76	1.82
Number of samples					38	38	38	1	37	38

NC Not calculated.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-10. - Subarea BIII--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
404	10.000	140.000			1.34	1.16	0.24	0.06	29.90	5.50
407	11.810	137.405	>50	16.00	1.68	1.25	.23		29.20	4.29
408	11.847	137.445	>50	10.00	1.66	1.20	.25		29.50	4.68
409	11.803	137.438	>50	16.30	1.66	1.20	.24		29.10	4.59
410	11.810	137.472	<20	.80	1.63	1.47	.20		29.20	3.57
411	12.183	137.683		11.30	1.54	1.24	.22		29.40	5.23
412	12.160	137.707	>50	15.60	1.56	1.20	.21		27.30	4.74
413	12.173	137.735		1.10	1.60	1.40	.30		26.90	4.64
414	12.200	137.745	<20	.20	1.57	1.44	.18		26.30	3.79
415	12.148	137.743	>50	18.70	1.65	1.21	.22		29.10	5.03
416	12.142	137.767	<20	2.60	1.46	1.36	.20		26.50	4.93
417	11.738	138.353	<20	.05	1.51	1.45			28.50	3.76
418	11.732	138.370	<20	4.00	1.50	1.50	.23		27.40	5.04
419	11.722	138.373	<20	.04	1.70	1.81			27.50	4.53
420	11.272	139.070	<20	1.90	1.61	1.20	.20		30.20	4.51
421	11.248	139.090	<20	.09	1.29	1.50			29.40	3.59
422	11.228	139.165	<20	.03	1.27	1.45			28.10	3.73
423	11.608	139.128	<20	4.50	.91	1.53			21.60	4.82
424	11.728	139.137	<20	.04	1.40	1.48			28.60	3.68
425	11.718	139.148	<20	.03	1.32	1.46			26.60	3.33
426	11.707	139.180	20-50	9.50	1.44	1.14	.18		28.60	4.90
427	11.693	139.183	20-50	.90	1.91	1.54	.20		27.80	4.32
428	10.750	139.400			1.36	.68	.30		20.80	7.40
429	10.672	139.953			1.55	1.26	.23		30.70	5.22
430	12.145	137.757	37.1	9.04						
431	11.257	139.015		2.34						
432	11.247	139.068	<20	3.00						
433	10.650	139.100	20-50							
434	10.600	139.425	<20							
435	11.703	138.390	<20							
436	11.675	138.432	<20							
437	11.258	139.055	NOb							
438	11.008	139.988		15.00	1.74	1.25			32.10	3.31
439	11.017	139.967	<20		1.13	.73	.24		22.10	7.50
440	11.057	139.997			1.58	1.10	.18		30.10	5.20
441	11.067	139.997	<20	5.35	1.52	1.40	.24		28.30	4.40
442	11.067	139.998	<20	3.95						
443	11.083	140.000		5.62	1.50	1.29			27.40	3.23
444	11.117	140.000			1.52	1.40	.24		28.70	4.90
445	11.048	140.047	<20	.20	1.71	1.49	.19		30.80	4.00
446	11.048	140.045	<20	.31						
447	11.012	140.067			1.37	1.25	.21		24.70	4.30
448	11.043	140.070			1.19	.86	.30		21.50	6.08
449	11.037	140.078			1.35	1.11	.27		28.40	5.85
450	11.012	140.080			1.41	1.29	.19		29.40	4.50
451	11.095	140.082		4.32	1.72	1.34	.25		33.30	4.75
452	11.093	140.082		.73	1.85	1.36	.29		30.70	4.60
453	11.035	140.088			1.03	.96	.26		25.30	6.28
454	11.035	140.090			1.46	1.12	.26		25.90	5.88
455	11.010	140.092	<20	4.27	1.12	1.15	.17		22.80	6.00

See footnote at end of table.

TABLE B-10. - Subarea BIII--location, abundance, and analytical data--Continued

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
456	11.008	140.092	<20	0.10						
457	11.010	140.088			1.40	1.37	0.19		28.00	4.20
458	11.012	140.088			1.46	1.43	.21		27.10	4.20
459	11.013	140.088			1.07	.98	.16		20.00	4.20
460	11.012	140.088			1.10	1.23	.32		19.20	5.10
461	10.982	140.092			1.43	1.29	.27		28.50	4.62
462	10.980	140.092			1.41	1.03	.34		26.50	6.64
463	11.010	140.095			1.43	1.28	.26		24.80	5.55
464	11.013	140.095			1.20	1.62	.48		21.70	6.50
465	11.008	140.095			1.44	1.25	.25		28.30	5.30
466	10.970	140.097			1.40	1.29	.20		29.30	4.55
467	10.967	140.097			1.63	1.42	.19		28.80	3.83
468	11.025	140.102	<20	.62	1.49	1.50	.21		30.80	4.30
469	11.013	140.105			1.56	1.17	.26		27.50	5.43
470	11.017	140.115			1.49	1.12	.30		27.00	5.95
471	11.015	140.118			1.59	1.29	.21		27.70	4.72
472	11.015	140.118			1.55	1.23	.29		28.60	5.34
473	10.927	140.135			1.64	1.29	.22		28.80	4.87
474	10.927	140.135			1.75	1.33	.27		29.60	5.40
475	10.965	140.148			1.63	1.26	.25		28.80	5.19
476	10.950	140.158			1.03	.63	.50		23.40	10.45
477	10.990	140.182	<20	.77	1.87	1.22	.36		30.60	5.75
479	10.990	140.193	20-50	5.17	1.63	.97	.38		27.80	7.70
480	10.990	140.193	20-50	5.61	1.55	.91	.38		26.50	7.64
481	11.073	139.982		9.09						
482	11.050	139.983		6.87						
483	11.010	139.990		5.62						
484	11.073	139.992		4.65						
485	11.053	139.992		9.27						
486	11.023	139.998		2.24						
487	11.093	139.995		5.81						
488	11.047	140.000		2.17						
489	11.067	140.000	NOb							
490	11.050	140.000		4.25						
491	11.050	140.000		12.12	1.47	1.37	.16		30.30	4.00
492	11.023	140.000		2.20						
493	10.950	140.000		6.37						
494	10.917	140.000		.12						
495	11.100	140.017		2.00						
496	11.100	140.017		.60	1.45	1.40	.17		26.40	4.70
497	11.100	140.017		1.12	1.35	.93			22.90	9.95
498	11.067	140.017		12.75	1.44	1.07	.20		30.20	5.00
499	11.075	140.002		4.30						
500	11.065	140.002		4.90						
501	11.055	140.002		3.70						
502	11.040	140.003		4.80						
503	11.022	140.004		6.40						
504	11.028	140.004		3.90						
505	11.043	140.012		5.20						
506	11.042	140.022		7.30						

See footnote at end of table.

TABLE B-10. - Subarea BIII--location, abundance, and analytical data--Continued

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
507	11.044	140.038		6.00						
508	11.047	140.060		6.20						
509	11.018	140.083		1.00						
510	11.035	140.116		4.50						
511	11.034	140.146		8.10						
512	10.983	140.163		6.90						
513	9.217	139.700			1.28	.98	0.20		22.70	5.40
514	9.100	139.750			1.32	1.14	.38		22.80	6.65
515	8.850	139.833			1.41	1.35	.25		28.90	5.42
516	9.000	139.850			1.36	1.17	.21		29.80	4.50
517	8.783	139.883			1.28	1.32	.20			11.60
518	8.783	139.883			.94	.94	.26		16.80	6.70
519	8.950	139.883			1.47	1.46	.20	0.07	22.90	5.04
520	8.833	140.300			1.00	.45	.12		30.10	1.78
		Mean..................		5.27	1.45	1.24	0.25	0.07	27.20	5.18
		Standard deviation....		NC	0.21	0.22	0.07	NC	2.98	1.37
		Number of samples.....		74	75	75	75	1	74	75

NC Not calculated.
NOb No nodules observed in photographs or no nodules recovered in samples.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-11. - Study area B--location, abundance, and analytical data outside subareas

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
400	8.983	137.683			0.11	0.11	0.05		2.50	8.20
401	9.950	137.783			1.54	1.15	.19	0.08	33.00	4.08
402	9.943	137.802			1.54	1.15	.18	.08	24.24	4.97
403	8.350	138.783			.62	.47	.14			
405	13.667	137.500		2.93						
406	11.717	142.800		4.30						
521	13.033	139.083			1.73	1.51	.38		28.90	
522	13.033	139.083			1.24	1.09	.34		24.30	6.50
618	12.283	140.430			1.56	1.22	.19		29.90	4.43
619	10.983	142.617			1.23	.96	.31	.03	17.00	6.40
620	10.012	143.312		1.25	1.51	1.34			29.20	4.53
621	10.010	143.317		18.12	1.52	1.12	.25		31.80	4.60
622	10.008	143.320		21.25	1.49	1.07	.18		30.10	5.10
		Mean..................		9.57	1.28	1.02	0.22	0.06	25.09	5.42
		Standard deviation....		NC	0.49	0.40	0.10	0.03	9.22	1.34
		Number of samples.....		5	11	11	10	3	10	9

NC Not calculated.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-12. - Subarea CI--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m^2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
662	16.008	124.993	NPr							
663	14.983	125.000		13.88	1.32	1.26	0.23		28.20	5.62
664	15.067	125.000		20.00	1.41	1.21	.28	0.07	28.80	6.75
665	14.967	125.017			1.40	1.17	.22		29.90	5.60
666	16.033	125.017			1.46	1.13	.26	.07	28.40	7.15
667	15.255	125.025	>50	13.70						
668	14.933	125.067		12.50	1.38	1.05	.30	.07	28.60	7.00
669	15.033	125.067		15.62	1.43	1.15	.29	.07	28.20	7.05
670	15.000	125.067		27.75	1.42	1.19	.29	.07	27.70	7.13
671	15.067	125.083	NPr							
680	15.110	125.932	NPr							
681	15.230	125.943	>50	10.00						
682	14.765	125.977	NPr							
683	15.262	126.000			1.37	1.14	.26	.06	29.09	5.99
684	14.257	126.023			1.38	1.20	.26	.06	27.30	6.26
685	15.243	126.028	>50	15.00						
686	15.242	126.047	>50	29.00						
687	15.135	126.063	>50	17.00						
688	15.242	126.065	20-50	15.00						
689	15.130	126.078	>50	16.00						
692	15.135	126.092	>50	13.00						
693	15.140	126.105	20-50	11.00						
695	15.192	126.112	NPr							
699	15.778	126.187	NPr							
702	14.908	126.622			.58	.36	.15		14.40	11.06
703	14.045	126.663	NOb							
704	16.012	126.772			1.20	.78			25.10	6.92
710	15.000	125.000			1.22	1.31	.15		29.70	5.25
711	14.967	125.000			1.38	1.27	.25		28.60	6.30
712	14.967	125.000			1.40	1.17	.22		29.90	5.60
713	15.033	125.083			1.45	1.33	.22		27.90	6.35
735	15.104	125.950	>50	11.80						
736	15.252	125.987	20-50	4.60						
737	15.220	125.920	>50	9.12						
738	15.241	126.040	>50	15.20						
739	15.137	126.090	>50	12.20						
740	15.231	125.974		4.90	1.34	1.21	.24		28.40	6.16
741	15.242	126.043		8.84	1.37	1.10	.28		27.50	6.68
742	15.257	125.988		1.79	1.36	1.31	.24		28.20	5.68
743	15.243	126.026		6.51	1.29	.83	.28		22.60	7.54
744	15.217	125.932		8.93	1.33	.95	.28		26.10	7.02
745	15.182	125.893		12.35	1.39	1.10	.36		28.10	6.49
746	15.205	125.868		9.95	1.19	.84	.31		26.20	8.09
747	15.207	125.953		8.48	1.26	.80	.30		26.20	8.16
748	15.188	126.025			1.34	1.02	.22		26.30	6.25

See footnote at end of table.

TABLE B-12. - Subarea CI--location, abundance, and analytical data--Continued

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
749	15.190	126.037		6.49	1.18	0.97	0.24		27.50	6.37
750	15.195	126.065		13.81	1.26	1.06	.29		27.20	6.06
751	15.193	126.098		11.44	1.26	.81	.29		27.20	6.06
752	15.137	126.107		10.21	1.33	.89	.27		26.60	6.92
753	15.137	126.062		15.23	1.32	.92	.28		27.00	6.32
754	15.140	126.045		8.81	1.31	.87	.24		25.70	6.63
755	15.132	125.990		9.24	1.23	.60	.36		23.80	9.51
756	15.138	125.955		16.84	.95	.47	.31		15.70	8.87
757	15.105	125.957		10.29	1.04	.46	.34		23.40	10.34
758	15.140	126.015			1.30	.94	.23		25.10	6.17
759	15.098	126.035		16.06	1.17	.73	.24		19.90	6.27
760	15.108	126.047		12.15	1.26	.93	.26		26.50	6.15
761	15.000	125.433			1.00	.82	.38		22.20	9.70
762	15.067	125.000		20.00	1.54	1.41	.26		31.10	6.29
763	14.983	125.000		15.00	1.46	1.40			29.20	5.64
764	14.967	125.000		11.88	1.53	1.36	.26		29.40	6.08
765	14.933	125.067		15.00	1.41	1.09	.27		29.90	6.59
766	15.033	125.067		17.50	1.49	1.21	.27		29.70	6.76
767	15.050	125.083			1.43	1.20	.25		29.50	5.78
768	15.033	125.083			1.51	1.26	.24		30.60	5.29
769	15.000	125.000			1.66	1.09	.24		25.00	6.22
770	15.760	126.008			1.19	1.08			26.50	5.60
771	15.245	126.535			1.41	1.09	.25		25.70	6.47
772	15.778	126.187			1.51	1.26	.27		28.80	5.84
773	15.297	125.922		9.80	1.38	.99	.32		27.30	8.30
774	15.297	125.473		2.60	1.43	.98	.29		27.20	8.09
775	15.343	125.447		7.70	1.33	1.21	.30		26.30	6.30
776	14.217	125.487			1.46	1.16	.19		28.10	5.50
777	15.132	125.132			1.21	.95	.18		25.20	7.82
778	15.043	125.138			1.39	1.08	.23		29.80	7.47
779	15.150	125.142			1.38	1.07	.22		31.10	7.21
780	15.148	125.162			1.04	.56	.37		26.00	12.80
781	15.100	125.167			1.17	.84	.21		25.10	7.58
782	15.000	125.067		28.75	1.45	1.19	.25		31.10	6.60
783	15.778	126.187			1.50	1.36	.26		26.60	5.96
784	15.233	126.500			1.51	1.05	.27		27.20	6.86
785	16.025	125.672		9.21						
786	16.083	124.967		4.25						
787	14.242	124.975			1.41	1.12	.28		26.50	6.75
788	16.050	124.983		3.50	1.02	.76			20.60	7.75
789	16.017	124.983		1.88	1.58	1.17			28.20	6.15
790	15.033	125.000		1.25						
791	15.255	125.025		11.50						
792	14.967	125.067		.62						
793	15.067	125.083			1.30	1.24	.23		23.20	5.95

See footnote at end of table.

TABLE B-12. - Subarea CI--location, abundance, and analytical data--Continued

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
794	15.340	125.902	<20							
795	15.327	125.907	<20							
797	15.758	126.167		5.47						
798	15.275	125.523		10.94						
799	15.273	126.867		14.11						
800	15.277	126.153		8.88						
801	14.292	126.257		14.67						
		Mean..................		11.67	1.33	1.04	0.26	0.07	26.79	6.91
		Standard deviation....		NC	0.17	0.23	0.04	0.00	3.23	1.40
		Number of samples.....		59	64	64	60	7	64	64

NC Not calculated.
NOb No nodules observed in photographs or no nodules recovered in samples.
NPr Nodules present, no additional information available.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-13. - Subarea CII--location, abundance, and analytical data

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m²	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
650	14.546	117.270			1.40	0.92	0.17		28.30	9.60
651	14.557	117.303			1.26	.95	.10		27.30	9.40
652	14.557	117.303			1.45	.96	.19		28.20	8.40
653	14.533	117.320			1.23	.87	.26		24.60	10.40
654	14.530	117.327			1.40	.91	.19		26.60	8.70
655	14.527	117.353			1.51	.93	.17		29.20	8.30
656	14.612	117.360			1.37	1.01	.14		29.60	7.20
706	14.438	117.152			1.41	1.16	.16		28.90	6.63
707	14.207	118.875			1.18	.99	.21		23.70	7.26
715	14.970	116.228			1.42	.90	.11	0.06	31.00	6.66
718	14.767	116.933	<20		1.43	.97	.14		28.50	7.20
719	14.433	117.200			1.89	1.06	.08	.08	26.80	10.30
720	14.530	117.258	NPr							
721	14.553	117.287			1.45	1.02	.20	.08	29.80	8.00
722	14.577	117.310			1.52	1.02	.17	.06	28.50	8.00
723	14.587	117.337			1.14	.72	.18	.06	24.80	10.85
724	14.573	117.367	NPr							
727	14.585	118.555			1.24	.90	.18	.06	26.90	8.80
		Mean.............................			1.39	0.96	0.17	0.07	27.67	8.48
		Standard deviation................			0.18	0.10	0.04	0.01	2.02	1.34
		Number of samples.................			16	16	16	6	16	16

NPr Nodules present, no additional information available.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

TABLE B-14. - Study area C--location, abundance, and analytical data outside subareas

Index No.	North latitude	West longitude	Population,[1] pct	Abundance, kg/m2	Analysis, wt pct, dry basis					
					Ni	Cu	Co	Mo	Mn	Fe
657	15.093	120.748			1.26	1.13	0.18		27.60	6.90
658	14.585	121.062			1.43	1.01	.21		28.80	6.90
659	14.603	121.032			1.32	1.03	.22		27.30	6.50
660	14.247	123.703			1.39	1.19	.23		28.10	6.30
661	14.917	124.200			1.22	1.39	.35	0.04	22.50	7.93
672	13.052	125.480			.98	1.03	.21	.04	25.00	4.35
673	12.327	125.500	NPr							
674	12.323	125.500		9.25	1.40	1.15	.13	.08	29.20	6.43
675	12.342	125.500	NPr							
676	12.337	125.502	NPr							
677	12.330	125.507	NPr							
678	12.327	125.507	NPr							
679	16.583	125.583	20-50		1.16	.72	.30		22.00	8.60
690	13.683	126.083	NPr							
691	13.680	126.083	NPr							
694	13.755	126.112			1.37	.95			28.10	7.26
696	13.757	126.118			1.41	1.20			29.10	5.88
697	13.760	126.118	NPr							
698	13.762	126.123	NPr							
700	13.780	126.228	20-50							
701	13.773	126.228	20-50							
705	13.257	126.800		14.11	1.21	1.01	.20	.06	22.80	6.08
708	13.905	120.813			1.40	1.16	.27		28.20	7.06
709	12.530	122.458			1.29	1.39	.19		29.80	5.20
714	12.275	120.158			.62	.42	.42		22.00	19.80
716	13.673	122.307			1.11	1.05	.25	.05	22.70	7.65
717	13.765	116.630								
725	11.233	117.483		10.66	1.46	1.23			31.70	6.39
726	13.450	118.417			.99	1.10	.55		31.50	4.20
728	11.427	118.757	NPr							
729	14.478	119.318	20-50	16.40						
730	14.837	119.663								
731	13.500	119.700								
732	14.835	120.142								
733	12.330	122.288			1.40	1.45	.15		34.90	4.32
734	12.298	122.418			1.08	1.06	.20		25.80	4.90
796	13.750	126.183	<20							
		Mean................		12.61	1.24	1.09	0.25	0.05	27.22	6.98
		Standard deviation...		NC	0.21	0.23	0.11	0.02	3.65	3.34
		Number of samples....		4	19	19	16	5	19	19

NC Not calculated.
NPr Nodules present, no additional information available.
[1]Amount of seafloor covered with nodules.

NOTE.--Blank indicates no information available.

50 84

TN295
.U4
2933

HECKMAN
BINDERY INC.
FEB 84

FIGURE 7. - Station locations, nodule occurrences, and grades in study area B. Topography adapted from Heezen (25).

TN295
.U4
1933

HECKMAN
BINDERY INC.
FEB 84

FIGURE 5. - Station locations, nodule occurrences, and grades in study area A. Topography adapted from Heezen (25).

TN295
.U4
1933

HECKMAN
BINDERY INC.
FEB 84
N. MANCHESTER,
INDIANA 46962

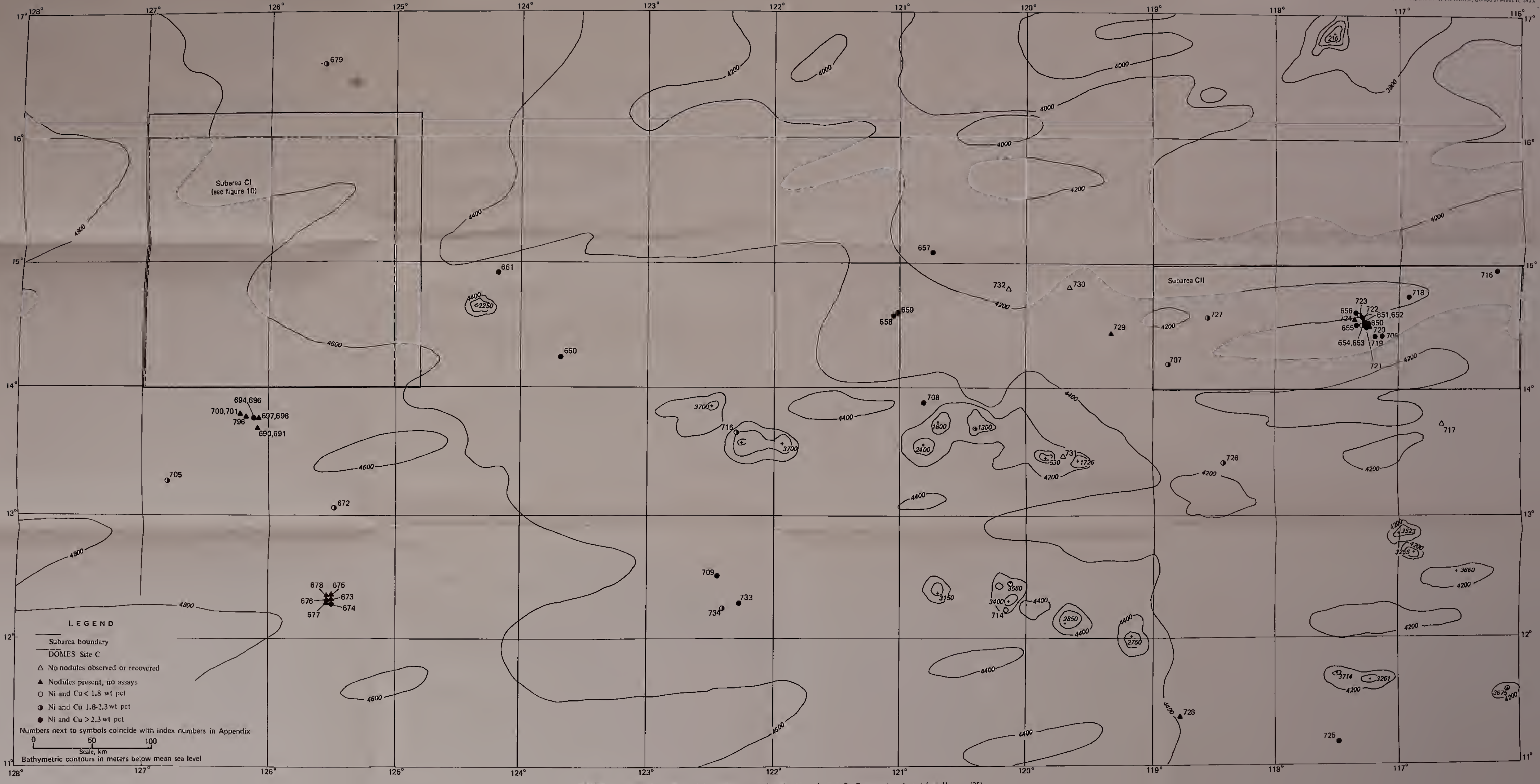

FIGURE 9. - Station locations, nodule occurrences, and grades in study area C. Topography adapted from Heezen (25).

LIBRARY OF CONGRESS
0 002 959 891 5